UNITED STATES BUREAU OF MINES
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ORE-DRESSING BIBLIOGRAPHY, 1931-32

This paper presents the results of work done under a cooperative agreement between the United States Bureau of Mines and the University of Utah.

BY

T. H. MILLER AND R. L. KIDD

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By T. H. Miller[2] and R. L. Kidd[3]

(In cooperation with the University of Utah.)

INTRODUCTION

An attempt has been made in this publication to present annotated references to all articles pertaining to ore dressing that were published during 1931 and 1932; in a few instances reference is made to articles published prior to 1931 where the articles have not been included in previous bibliography appended to the annual statistical discussions of ore concentration published by the Bureau of Mines. Articles have been taken from more than 75 periodicals, journals, serials, etc., and 423 articles are abstracted herewith. A number of these abstracts, particularly those appearing in foreign publications, have been taken from Chemical Abstracts (published by the American Chemical Society, Easton, Pa.), the general use of which the authors wish to acknowledge. Other annotations were taken directly from the original publications.

The entire field of ore dressing has been included in the bibliography; and articles dealing with crushing, grinding, classification, concentration, thickening, and filtration have been included, as well as on amalgamation, cyanidation, leaching, magnetic concentration, washing, etc., and on theory and laboratory operation. The articles are arranged in alphabetical order by authors. Lack of printing funds prohibited the inclusion of a subject index.

No attempt has been made in this report to cover the field of ore-dressing patents. Subsequent reports of this series may include this subject.

The following abbreviations were used:

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[1] The Bureau of Mines will welcome reprinting, provided that the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6734."

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Publications	Abbreviations
American Association for the Advancement of Science, New York.	Science
American Ceramic Society, Columbus, Ohio	Jour. Am. Ceram. Soc.
American Chemical Society, Washington, D. C.	Jour. Am. Chem. Soc.
American Cyanamide Co., Technical Department, New York.	Am. Cyan. Co., Tech. Paper.
American Institute of Mining and Metallurgical Engineers, New York.	
Transactions	Trans. Am. Inst. Min. and Met. Eng.
Technical Publications	Am. Inst. Min. and Met. Eng., Tech. Pub.
Mining and Metallurgy	Min. and Met.
Analele menelor din Romania, Bucharest, Rumania.	Analele menelor Romania
Australasian Institute of Mining and Metallurgy, Melbourne, Australia.	Proc. Australasian Inst. Min. and Met.
Black Hills Engineer, Rapid City, South Dakota.	Black Hills Eng.
Blast Furnace and Steel Plant, Pittsburgh, Pennsylvania.	Blast Furnace and Steel Plant.
Canadian Chemistry and Metallurgy, Toronto, Canada.	Canadian Chem. and Met.
Canada, Department of Mines, Mines Branch, Ottowa, Canada	
Annual Reports	Canadian Dept. Mines, Mines Branch, Ann. Repts.
Memorandum Series	Canadian Dept. Mines, Mines Branch, Mem. Ser.
Canadian Mining and Metallurgical Bulletin, Montreal, Canada.	Canadian Min. and Met. Bull.
Canadian Mining Journal, Toronto, Canada.	Canadian Min. Jour.
Cement and Cement Manufacture, London, England.	Cement and Cement Manuf.
Ceramic Society (England), Stoke-on-Trent, England.	Trans. Ceram. Soc. (England)
Chemische Apparatur, Leipzig, Germany.	Chem. App.
Chemical Engineering and Mining Review, Melbourne, Australia.	Chem. Eng. and Min. Rev.
Chemische Fabrik, Berlin, Germany.	Chem. Fabrik.
Chemical and Metallurgical Engineering, New York.	Chem. and Met. Eng.
Chemical, Metallurgical, and Mining Society of South Africa, Johannesburg, South Africa.	Jour. Chem. Met. and Min. Soc., South Africa.

Colliery Guardian, London, England.	Coll. Guard.
Colorado School of Mines Magazine, Golden, Colorado.	Colorado Sch. Mines Mag.
Combustion (Intern. Comb. Eng. Co.), New York.	Combustion.
Crushing and Grinding, London, England.	Crushing and Grinding.
Dansk Tidsskrift for Farmaci, Copenhagen, Denmark.	Dansk Tids. Farm.
Electrochemical Society, New York.	Trans. Electrochem. Soc.
Engineering and Mining Journal, New York.	Eng. and Min. Jour.
Faculty of Engineering, Tokyo Imperial University, Tokyo, Japan.	Jour. Faculty Eng., Tokyo Imp. Univ.
Génie civil, Paris, France.	Génie civil.
Glückauf Berg- und huttenmannische Zeitschrift, Essen, Germany.	Glückauf.
Industrial and Engineering Chemistry, Washington, D. C.	Ind. Eng. Chem.
Institution of Mining Engineers, London, England.	Trans. Inst. Min. Eng.
Institution of Mining and Metallurgy, London, England.	Bull. Inst. Min. and Met.
Internationale Bergwirtschaft und Bergtechnik, Halle, Germany.	Intern. Bergwirt. Bergtech.
International Association (New) for the Testing of Materials, Zurich, Switzerland.	New Internat. Assoc. Test. Mat.
Japanese Ceramic Society, Tokyo, Japan.	Jour. Japanese Ceram. Soc.
Jernkontorets Annaler, Stockholm, Sweden.	Jernkontorets Ann.
Journal of Physical Chemistry, New York.	Jour. Phys. Chem.
Kolloid-Beihefte, Dresden-Blasewitz, Germany.	Kolloid-Beihefte.
Kolloidchemische Beihefte, Dresden-Blasewitz, Germany.	Kolloidchem. Beihefte.
Kolloid-Zeitschrift, Dresden-Blasewitz, Germany.	Kolloid Ztschr.
Metal Industry (London), London, England.	Metal Ind. (London).
Metall und Erz, Halle, Germany.	Metall u. Erz.
Mineral Industry, New York.	Mineral Ind.
Mineralnoe Suire, Moscow, Russia.	Miner. Suire.
Mining Congress Journal, Washington, D. C.	Min. Cong. Jour.
Mining Journal (London), London, England.	Min. Jour. (London).
Mining Journal (Arizona), Phoenix, Arizona	Arizona Min. Jour.
Mining Magazine (London), London, England.	Min. Mag. (London).
Missouri School of Mines and Metallurgy, Rolla, Missouri.	Missouri Sch. Mines and Met., Tech. Ser.
Mitteilungen aus dem Kaiser-Wilhelm Institut fur Eisenforschung zu Dusseldorf, Dusseldorf, Germany.	Mitt. Kaiser-Wilhelm Inst. Eisenforsch, Dusseldorf.
Nature, London, England.	Nature.
Physics, Minneapolis, Minnesota.	Physics.

Physikalische Zeitschrift, Leipzig, Germany.	Physikal. Ztschr.
Proceedings of the World Engineering Congress, Tokyo, Japan.	Proc. World Eng. Cong., Tokyo.
Przemyśl Chemiczny, Warsaw, Poland.	Przemysl Chem.
Revue de metallurgie, Paris, France.	Rev. Metal.
Rock Products, Chicago, Ill.	Rock Products.
School of Mines of Western Australia, Kalgoorlie.	Sch. Mines West. Australia, Bull.
South African Mining and Engineering Journal, Johannesburg, South Africa	S. African Min. and Eng. Jour.
Stahl und Eisen, Pittsburgh, Pennyslvania	Stahl u. Eisen.
Tidskrift for Kjemi og Bergvesen, Oslo, Norway.	Tids. Kjemi Bergvesen.
Tzvetnuie Metallui, Moscow, Russia.	Tzvetnuie Metallui.
United States Department of Commerce, Bureau of Mines, Washington, D. C.	
Bulletins	Bull. Bureau of Mines.
Economic Papers	Ec. Paper, Bureau of Mines.
Information Circulars	Inf. Circ., Bureau of Mines.
Reports of Investigations	Repts. of Investigations, Bureau of Mines.
Technical Papers	Tech. Paper, Bureau of Mines.
Zement, Charlottenburg, Germany.	Zement.
Zeitschrift fur physikalische Chemie, Abt. A., Leipzig, Germany.	Ztschr. physikal. Chem.

1. ADAMS, A. S., KOBEY, S. M., and SAYERS, M. J. Flotation of hematite a laboratory success. Eng. and Min. Jour., vol. 132, 1931, pp. 54-55.

   Laboratory flotation tests of Mesabi log-washer tailings containing 17 percent iron in the form of hematite indicate a recovery of 74 percent of the iron in concentrates containing 59 percent iron at a cost of $1.40 a ton. Fine grinding is necessary; and the flotation reagents used were sodium oleate, 1 pound per ton, and sodium carbonate, 2 pounds per ton. Sodium silicate was an aid up to 1/2 pound per ton but decreased the grade of concentrates when used in larger quantities. Optimum operating conditions were determined.

2. AGNEW, JOHN A. Flotation of gold ores. Canadian Min. Jour., vol. 52, 1931, pp. 662-664.

   Describes the new treatment plant of Wiluna Gold Mines (Ltd.) at Wiluna, West Australia. The plant has a capacity of 40,000 tons a month and employs straight flotation, followed by roasting and cyanidation of the concentrates. Flotation recovers 92 percent of the gold in the ore and cyanidation 90 percent of the gold in the concentrates. Flotation reagents used are ethyl xanthate and pine oil (or eucalyptus oil). The flow sheet of the reduction plant is given.

3. ______ The flotation of gold ore. Canadian Min. Jour., vol. 52, 1931, pp. 710-711.

   Discusses the milling plant at Lake View and Star, which has a capacity of 15,000 tons a month and treats gold ores from several mines near Kalgoorlie, West Australia. The flow sheet is given. About 35 percent of the gold is recovered on corduroy strakes and melted into bars. The pulp is then treated by flotation, and the concentrates are roasted and cyanided. Metallurgical recovery is somewhat lower than that obtained by roasting and cyaniding the raw ore, but operating costs are much lower, resulting in a higher economic recovery.

4. AHLFELD, FREDRICH. Occurrence and recovery of gold in the Bolivian Andes. Metall u. Erz, vol. 28, 1931, pp. 163-167.

   Describes geology of gold deposits in Bolivia. The gold-bearing material is hydraulicked and amalgamated.

5. ______ Basic principles in the dressing of Bolivian tin ores. Metall u. Erz, vol. 29, 1932, pp. 178-181.

   Outlines treatment of sulphide, oxide, and silver-bearing tin ores and gives their distribution in Bolivia.

6. ALEINIKOV, N. A. Residual oil from rectification of fusel oil as a flotation reagent. Miner. Suire, no. 5, 1930, pp. 1287-1291.

   The dark, heavy residual oil from rectification of fusel oil was tested as a reagent in flotation of quartzitic copper sulphide ores. The oil produces abundant, stable froth and gives a very satisfactory extraction of copper from the ore. Mixture of the oil with ethyl xanthate gives still better results and permits considerable saving of oil and xanthate.

7. ALLEN, A. W. Physical and mechanical aspects of the cyanide process. Trans. Electrochem. Soc. (Prepr. 60), 1931, 5 pp.
Describes effect of supergrinding of a gold ore in the presence of an unstable solvent; author points out that mechanical operations are of great importance in the effective solution of gold in cyanide. The author recommends: (1) Preliminary milling in lime water; (2) dewatering and thickening of the pulp; (3) cyanide treatment of the pulp or sand; and (4) regeneration of cyanide.

8. ALLEN, A. W., and LEAVER, E. S. The cyanidation of complex ores. Chem. Eng. and Min. Rev., vol. 24, 1931, pp. 41-44.
Discusses complex ores, precipitation of precious metals by zinc dust, and present status of cyanidation.

9. AMBROSE, H. A., and LOOMIS, A. G. Some colloidal properties of bentonite suspensions. Physics, vol. 1, 1931, pp. 129-136.
The emulsoid colloids in bentonite respond to changes in pH of the dispersion medium by changes in rate of settling, viscosity, and swelling. The curve showing the relation between pH and percent of sedimentation in a definite time has a maximum at the isoelectric point, with minima on each side corresponding to points of maximum absorption. As pH is increased or decreased further, flocculation begins with increased rate of settling. The viscosity curve is closely related to the rate of settling; maximum swelling occurs at approximately the point of maximum viscosity. Such salts as sodium chloride and potassium chloride are markedly effective in preventing the swelling of bentonites.

10. ANDERSON, A. K. A recovery of gold from the ore of the Jackson-Manion Mines (Ltd.), Woman Lake, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 9-13.
Cyanidation yields a high recovery of the gold with fine grinding, using 6 to 7 pounds lime per ton and a cyanide solution of 1 to 2 pounds cyanide per ton.

11. _____ Concentration of copper-nickel-bearing gabbro from Engineers Holding Co., Michipicoten, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 6-8.
The ore analyzed 1.23 percent copper, 0.51 percent nickel, 0.14 percent zinc, 0.01 ounce gold, 0.25 ounce silver, and 0.06 ounce platinoids (chiefly palladium) per ton. A bulk flotation yields high recoveries of copper (98 percent), a large percentage of nickel (76.5 percent), and the major amounts of the precious metals (gold 59 percent, silver 96 percent, and platinoids 34 percent).

12. _____ The concentration of a lead-zinc-silver ore from the Regal Silver Mines (Ltd.), Albert Canyon, British Columbia. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 112-115.
Five samples of sulphides of lead, zinc, and iron in siliceous gangue assaying 4.02 to 11.67 percent lead, 1.20 to 3.7 percent zinc, and 3.88 to 10.70 ounces of silver per ton gave recoveries of over 90 percent of the silver and lead by flotation with soda ash, cyanide, and zinc sulphate in the ball mills, Aerofloat no. 25 or potassium xanthate in the lead cells, and Aerofloat no. 25 or xanthate, pine oil, and copper sulphate in the zinc cells.

13. ANDERSON, A. K. The concentration of silver in the ore of the Animikie Mines (Ltd.), Port Arthur, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 90-94.

Proustite in an ore of limestone and calcite gangue carrying a small amount of iron sulphide gave the ore an assay of 46.0 ounces of silver and 0.02 ounce of gold per ton. Flotation can be depended on to yield a recovery of 95 percent of the silver in concentrates (refloating a rougher product) assaying 2,000 ounces of silver per ton. The circuit was alkaline (5 pounds soda ash per ton); Aerofloat no. 25 was used at the rate of 0.20 pound per ton.

14. _____ The concentration of the copper ore of the Winsor mine, La Sarre, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 65-66.

A massive pyrite and pyrrhotite ore, with associated chalcopyrite and small quantities of sphalerite, assaying 4.70 percent copper, 1.78 percent zinc, 0.02 ounce gold, and 0.34 ounce silver to the ton, was readily concentrated by flotation. The use of 6.0 pounds soda ash, 0.20 pound cyanide, and 0.12 pound potassium xanthate per ton of ore recovered 96 percent of the copper in concentrates assaying 27.50 percent copper. A rougher concentrate assaying 40 percent zinc, with a recovery of 67 percent of the zinc, was made by using 0.08 pound xanthate, 0.02 pound pine oil, and 1.0 pound copper sulphate per ton in the zinc circuit.

15. _____ The flotation of copper and iron pyrites in the Aldermac mine, Rouyn area, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 4-12.

A sulphide ore assaying 1.98 percent copper, 0.49 percent zinc, 40.5 percent iron, 0.09 ounce gold, and 1.18 ounces silver per ton was floated to give copper concentrates, zinc concentrates, pyrite concentrates, and pyrrhotite tailings. The use of soda ash-cyanide, lime-cyanide, or lime as conditioning reagents gave copper recoveries of 89.7, 73.0, and 86.2 percent, respectively, in products assaying 14.75, 18.91, and 27.28 percent copper, respectively. For the most satisfactory test the following reagents were used: Ball mill--lime 6.0 pounds; copper circuit—potassium xanthate 0.06 pound, pine oil 0.04 pound; zinc circuit-potassium xanthate 0.05 pound, copper sulphate 1.0 pound, pine oil 0.04 pound; iron circuit—amyl xanthate 0.30 pound, sulphuric acid 5.0 pounds, pine oil 0.06 pound per ton. Acidifying the pulp after zinc flotation yields the highest recoveries of pyrite (50 to 60 percent) in a product containing 48 to 50 percent sulphur and 0.10 to 0.05 percent zinc. Dewatering the tailing from the zinc flotation before acidifying saves acid and does not affect recovery of pyrite.

16. ANDERSON, A. K. The recovery of copper and gold from the ore of the Tashota Gold Mines (Ltd.), Tashota, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 122-129.

A chalcopyrite-pyrrhotite ore assayed 1.57 percent copper, 1.85 ounces gold, and 1.87 ounces silver per ton. Results are given of tests with flotation, tabling, amalgamation, and cyanidation in various combinations. Tests indicated recovery of 90 to 95 percent of the copper. A combination of flotation, amalgamation, and cyanidation yields maximum recoveries. On account of the danger of oxidation of the ore before milling, resulting in a loss of copper in the tailing, and because copper sulphides are strong cyanicides, a cyanide treatment is not recommended in practice.

17. _____ The recovery of gold and silver from the ore of the Gem Lake mine, East Central Manitoba. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 58-64.

The quartzitic ore tested contained 3.5 ounces of gold and 0.35 ounce of silver a ton, with no arsenic, antimony, bismuth, or copper. Tests included amalgamation, cyanidation, and concentration, alone and in various combinations. Best results were obtained by amalgamation and cyanidation, yielding 98.5 percent of the gold and 97.3 percent of the silver.

18. _____ The recovery of gold from the ore of the Belledat-Goudreau mine, Goudreau, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 71-81.

The ore tested consisted of a quartz gangue carrying 1.92 ounces gold and 0.82 ounce silver to the ton associated with a little pyrite, galena, and graphite. A good recovery of gold from the crude ore by cyanidation was not possible, due to the graphite. About 80 percent of the gold was recoverable by amalgamation, and subsequent concentration raised the recovery to 98.9 percent, but in continuous operations the graphite coated the plates and the recovery was much lower. Grinding the ore with lime and kerosene followed by flotation with pine oil removed the graphite in concentrates containing 22.56 ounces gold and 25.2 ounces silver to the ton which could be treated by smelting. Cyanidation of the graphite-free tailings gave an over-all gold recovery of 98.9 percent.

19. _____ The recovery of gold from the ore of the British Canadian Mines (Ltd.), Mine Centre, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 13-17.

The ore consisted of quartz and basic rock containing small amounts of pyrite along fractures in the quartz and assaying 0.60 ounce gold per ton. Cyanidation gives a recovery of 99 percent with ore ground to pass 100-mesh and 22 percent on 150-mesh. At 65-mesh a recovery of 98.3 percent is obtained.

20. ANDERSON, A. K. The recovery of gold from the ore of the Casey Mountain Operating Syndicate, Summit Lake, Patricia Portion, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 150-154.
A magnetiferous quartz carrying gold and small amounts of mispickel, pyrite, and traces of galena and chalcopyrite assayed 0.72 ounce of gold and 0.11 ounce of silver per ton and 0.42 percent arsenic. Amalgamation gave 82 percent recovery, flotation 75 percent, and cyanidation 96.5 percent.

21. _____ The recovery of gold from the ore of the Evangeline Gold Mines (Ltd.), Liscomb Mills, Nova Scotia. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 69-71.
An ore containing free gold and about 2 percent arsenopyrite was treated by amalgamation and concentration. Amalgamation yielded 96.4 percent of the gold; and the over-all recovery, including the gold in the arsenopyrite concentrates, was 98.7 percent.

22. _____ The recovery of gold from the ore of the Night Hawk Lake Mining Co. (Ltd.), Night Hawk Lake, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 130-135.
Cyanidation of a porphyry containing pyrite assaying 0.14 to 0.17 ounce of gold and 0.03 to 0.07 ounce of silver per ton gave highest recoveries (97 percent) with a cyanide consumption of 0.60 pound per ton of ore. Results by flotation and amalgamation alone and in combination are given.

23. _____ The recovery of gold from the ore of the St. Anthony Gold Mines (Ltd.), Savant Lake, Ontario. Canadian Dept. Mines, Mine Branch, Rept. 720, 1931, pp. 84-86.
Amalgamation of a quartz ore ground to minus 48-mesh yielded 90.3 percent of the gold and 45.5 percent of the silver. At minus 35-mesh the gold recovery was 69.3 percent. Cyanidation of minus 48-mesh pulp recovered 90.7 percent of the gold and 77.3 percent of the silver.

24. _____ The recovery of gold from the tailings of the County Harbour Mines, Nova Scotia. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 65-69.

25. _____ The selective flotation of a lead-zinc ore from the Haslat-Duck Lake Mines (Ltd.), Schreiber, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 139-143.
The ore consists of massive, fine-grained intimately associated sulphides of lead, zinc, and iron in a siliceous gangue. It contained 4.94 percent lead, 8.05 percent zinc, 0.19 percent copper, 0.11 ounce gold, and 2.06 ounces silver per ton. The lead concentrates contained 34.93 to 45.60 percent lead, 0.46 to 0.65 ounce gold, and 12.49 to 16.19 ounces silver per ton and carried 90.6 to 95.4 percent of the lead, 48.9 to 67.2 percent of the gold, and 77.0 to 82.5 percent of the silver. The zinc concentrates assayed 27.14 to 44.51 percent zinc and carried 76.5 to 83.9 percent of the metal.

26. ANDERSON, A. K., and HALFORD, D. S. The concentration of the cobalt ores of the Aguanico Cobalt Mines (Ltd.), Cobalt, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 80-90.

An ore of cobaltite, smaltite, and erythrite in a diabase gangue with small amounts of calcite assaying 2.60 percent cobalt, 0.60 percent nickel, 0.23 ounce silver, and 0.02 ounce gold per ton was treated by flotation. By using 7 pounds soda ash and 0.05 Flotagen per ton in the ball mill and 0.10 pound Flotagen, 0.25 pound pine oil, and 1.0 pound copper sulphate per ton in the flotation machine a concentrate (11.93 percent by weight) assaying 17.76 percent cobalt and containing 83.67 percent of the cobalt, a middling (10.16 percent) of 1.54 percent cobalt containing 16.18 percent more of the cobalt, and a tailing (77.91 percent) assaying 0.33 percent cobalt were obtained. Flotation tests were made with samples from various parts of the present plant (using gravity separation), with varying success.

27. ANDERSON, A. K., GODARD, J. S., and PARSONS, C. S. The differential flotation of copper and iron sulphides in the ore of the Aldermac Mines (Ltd.), Aldermac, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 101-110.

An ore of pyrite and pyrrhotite with small amounts of chalcopyrite assaying 1.75 percent copper, 28.2 percent iron, and 39.3 percent sulphur was floated easily. To obtain maximum recoveries low-grade rougher concentrates are produced and cleaned. The copper minerals were sensitive to cyanide; if over 0.04 pound of cyanide per ton is added the copper is depressed. Recoveries fall off when grinding exceeds 65 percent solids. Typical results show 93.6 percent copper recovery in rougher concentrates assaying 20.72 percent copper and 63.2 percent of the mill feed recovered as pyrite concentrates assaying 52.2 percent sulphur, with the following reagents in pounds per ton of ore: For ball mill, soda ash 3.7, cyanide 0.035, Aerofloat no. 25 0.06; for copper cells, cresylic acid 0.036; for iron cells, soda ash 2.1, Aerofloat no. 25 0.09, copper sulphate 0.5, xanthate 0.66. The ball-mill discharge was 58 percent solids.

28. ANDERSON, C. S. Mining and milling in the Vermont copper district. Eng. and Min. Jour., vol. 131, 1931, pp. 208-210.

Describes milling operations of the National Copper Co. near Pompoonoosue, Vt. The plant, of about 400 tons daily capacity, treats ore containing quartz, feldspar, mica, pyrrhotite, and chalcopyrite by straight flotation. The ore treated in 1929 and 1930 averaged 2.11 percent copper, 0.32 percent zinc, 36.40 percent iron, and 23.02 percent sulphur. A recovery of 91.8 percent of the copper was made. Flotation reagents used were lime, pine oil, potassium amyl xanthate, thiocarbanilid, and arthotoluidin. Article gives mill flow sheets and metallurgical data.

29. ANDERSON, JOHS. Mesh openings and particle sizes. Zement, vol. 20, 1931, pp. 224-226 and 242-245.

The author points out the difficulties in making accurate sieve analyses due to irregularities in the sieves and other conditions of the test. For accurate results, the actual diameters of the particles should be calculated from the weight determinations and microscopic counts.

30. ANDERSON, JOSEPH T. The solubility of gold in mercury. Part V, Jour. Phys. Chem., vol. 36, 1932, pp. 2145-2165.
Describes electrically heated air bath, capable of temperature uniformity of 1° in the central portion at about 400°; also solubility tube with capillary filter, which permits detection of the disappearance of the solid phase of opaque solutions. Forty-two determinations of the solubility of gold in mercury were made between 280 and 400° by the synthetic method. Article gives methods of analysis of mixtures of gold, mercury, and lead and describes filtering tube which operates on the centrifuge principle. With this tube the composition of solid and liquid phases was determined for gold and mercury between 295 and 360°. Four runs were made with lead added to the gold and mercury; and on the assumption that all the lead is in the liquid phase, a corrected composition of the solid phase was calculated based upon the amount of lead found in the solid phase after filtering. The gold-mercury solubility curve shows a break at 310°. A phase diagram is given which seems to agree with the results of Parravano, Blitz, Meyer, and Pabst.

31. ANDERSON, R. A. Lake View and Star Gold Mines. Min. Mag. (London), July 1932, pp. 9-20.
Reviews in detail changes at the reduction plant, Kalgoorlie, West Australia. The plant was formerly a dry-grinding, all-roasting cyanide plant. The new plant, treating 30,000 tons of ore a month, uses 2-stage wet grinding and flotation followed by drying, roasting, and cyaniding of the flotation concentrates. Corduroy tables are used, following the open-circuit primary ball mills, which recover about 30 percent of the total gold. Flotation recovery exceeds 90 percent with a ratio of concentration of 10 into 1. New flow sheet is given.

32. ANDREASEN, A. H. M. Grain size and fineness. New Internat. Assoc. Test. Mat., September 1931, 6 pp.
The term "sorting value" is suggested for the grain size at which a sieve separates. Its ratio to the size of the mesh for ordinary substances is 0.8 to 0.9. A single number is insufficient to characterize the fineness of materials, consisting of particles of different sizes, and a curve such as the distribution curve must be used.

33. _____ Laws of fine grinding. Crushing and Grinding, vol. 1, 1931, p. 69.

34. _____ Technical analysis of fineness (particle size). Tek. Tid., Uppl. C, Kemi, vol. 61, 1931, pp. 65-68, 75-80.

35. _____ The fineness of grinding in industry. Tek. Tid., Uppl. C, Kemi, vol. 62, 1932, pp. 6-8, 12-15, and 20-24.
Discusses practical importance of knowing the fineness of materials employed for the various technical purposes, such as pigments, abrasives, cement, ceramics, ores, etc. Treats determination of the space between particles at certain particle sizes mathematically. Describes various methods and devices employed to produce materials of a certain fineness.

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36. ANDREASEN, A. H. M. The fineness of solid materials. Dansk. Tids., Farm., vol. 4, 1930, pp. 261-263.

The characteristics of solid materials change as the fineness of particle sizes increases. Particle sizes of 1-mm diameter can be readily distinguished, but if the size is reduced to 0.1 mm the material becomes "dusty". Particles of 0.01-mm diameter tend to lump together, and 0.001-mm particles feel "fatty" and adhere very readily to surfaces. The degree of fineness can be determined down to 0.06 mm by sieving, while below this limit the sedimentation method must be employed. In water, the range of measurement lies between 0.03 and 0.001 mm. With glycerol the upper limit can be increased to 0.3 mm. A pipette for the estimation of particle size is described. This method consists essentially in drawing a suspension of the material into a pipette, and at regular time intervals withdrawing definite volumes from which the solid material is determined by evaporating off the liquid. By application of Stokes law the particle size is determined graphically. Article discusses influence of grain size in technical operations, as well as methods for obtaining the desired degree of fineness.

37. ANDREWS, LEONARD. High-velocity hydraulic classification. Trans. Inst. Min. and Met. (London), vol. 37, 1928, pp. 406-436.

38. _____ Hydraulic classifiers as an aid to batch grinding. Trans. Ceram. Soc. (England), vol. 30, 1931, pp. 98-111.

39. ANDRUS, D. E. Mining methods and costs at the Montana mine concentrator of the Eagle-Picher Lead Co., Ruby, Ariz. Inf. Circ. 6497, Bureau of Mines, 1931, 14 pp.

The ore treated at this 250-ton straight flotation plant built in 1928 consists of galena, sphalerite, and pyrite, with minor quantities of chalcopyrite and tetrahedrite in a quartz gangue containing considerable colloidal material. The ore is crushed in a jaw crusher and rolls to about 5 mm and ground in a conical mill in closed circuit with a belt drag classifier. The pulp is treated in 8 Butchart flotation machines, 4 each in the lead and zinc circuits. Flotation reagents are sodium carbonate, lime, xanthate, thiocarbanilid, Aerofloat, cresylic acid, pine oil, sodium cyanide, zinc sulphate, sodium sulphite, copper sulphate, and sodium silicate. Lead concentrates containing 57.05 percent lead and zinc concentrates containing 52.36 percent zinc are produced with recoveries of 92.59 percent of the lead in the lead concentrates and 68.29 percent of the zinc in the zinc concentrates. The report gives flow sheets and cost and metallurgical data.

40. APPLEYARD, KENELM C. Dry cleaning of coal in England. Am. Inst. Min. and Met. Eng. Tech. Pub. 374, 1930, 34 pp.

41. APPLEYARD, KENELM C., and O'TOOLE, EDWARD. The present state of the pneumatic process for cleaning coal; I, II, and III. Blast Furnace and Steel Plant, vol. 18, 1931, pp. 1834-1835; vol. 19, 1931, pp. 268-272, 418-422.

42. ARMYTAGE, H. E. Flotation: Some operating details. Inst. Min. and Met. (London), Bull. 329, 1932, 14 pp.; Bull. 330, 1932, pp. 15-24; Bull. 331, 1932, pp. 27-28.
Discusses laboratory devices and methods for flotation research and control.

43. ASEEV, N. P., BELOGLAZOV, K. F., and GREIVER, N. S. Ammonia leaching of cupriferous sandstones. Tzvetnaie Metallurgie, 1931, pp. 703-712.
Gives results of experimental ammonia leaching of cupriferous sandstones from the Ural district. The ores contain 1.3 to 1.8 percent copper. The recovery was lower than that obtained by sulphuric acid leaching.

44. ________ Hydrometallurgical acid treatment of carbonate ores. Tzvetnine Mettaline, 1931, pp. 579-581.
An investigation of the process of solution of marble, calcite, and limestone in sulphuric acid of varying concentrations. The results may be briefly summarized as follows: (1) Formation of films of calcium sulphate leads to a sharply defined minimum of the speed of reaction of calcium carbonates at the concentration of about 2 percent sulphuric acid; (2) the thickness of the affected layer of carbonate depends on the concentration of the solution, but (for concentrations employed in practice) does not depend on the size of the material to be treated. The degree of crushing, therefore, has a great influence on the consumption of the reagent. (3) The results obtained give very important indications regarding their application to practical processes of extracting metals from carbonates, as well as any other material, in which the acid or any other reagent acting on a material forms an insoluble film. Experimental procedure and data are given.

45. AUSTRALASIAN INSTITUTE OF MINING AND METALLURGY. The development of processes for the treatment of crude ore, accumulated dumps of tailings, and slimes at Broken Hill, New South Wales. Proc. 80, 1930, pp. 379-444.
Describes development of the various processes which have led to present-day metallurgical practice at Broken Hill. These include blast furnacing, gravity concentration, magnetic separation, Potter process (use of hot, acidulated flotation solutions), Delprat process (flotation, using salt cake), Cattermole process (agglomeration in water suspension of selectively oiled particles), DeBavay process (film flotation), Elmore vacuum process (aeration during flotation by release of dissolved gases by vacuum), Minerals Separation process (flotation-refinement of Cattermole process), Murex magnetic process (differential application of magnetic coating, Henderson's process for oxidized lead, and differential flotation (Horwood, Lyster, Bradford (sodium chloride-sulphuric acid), Lowry and Greenway, Owen, and Bradford (copper sulphate) processes). Present practice with consumption of reagents, assays, and recoveries for each company is presented and trends in the field are discussed.

46. AVDEEV, K. A. Experiments on concentrating graphite shale from Ssojusnoje (Amur). Miner. Saire, no. 5, 1930, pp. 1134-1137.
The shale contained 1 .4 percent graphite. Flotation produced 46.2 percent concentrates, with a recovery of 83.3 percent of the carbon in the shale.

47. AVDEEV, K. A. Refining of graphitic schists from the town Soyuzruii on the Amur. Miner. Suire, no. 6, 1931, pp. 1022-1026.
The schist samples were powdered to 200-mesh and refloated 4 times with the addition of 3 to 4 Kg kerosene or pine oil to 1 ton of ore in an apparatus of the pneumatic type. The yield of concentrates was 14 percent; the tailings contained 4 percent graphite with an 80-percent extraction of graphite in concentrates containing 90 percent carbon.

48. AVERILL, CHARLES VOLNEY. Cyaniding low-grade gold ore. Chem. Eng. and Min. Rev., vol. 24, 1931, pp. 87-91.
Describes cyanide plant for treating gossan ore at the Mountain Copper Co. works, Shasta County, Calif. Gossan ore, hitherto considered not amenable to cyaniding, is treated successfully at the low cost for milling and mining of 85 cents per ton in a plant handling 550 tons per day. The standard Merrill-Crowe vacuum process of precipitating gold from solution is employed. Several tables and cost figures are given.

49. BADESCU, E. Application of flotation to the treatment of gold ores. Analele Minelor Romania, vol. 14, 1931, pp. 179-185, 239-242.
Briefly discusses flotation in the treatment of Rumanian gold ores.

50. BADESCU, EUGEN, and PROCKAT, FR. Effect of various xanthates on pure chalcopyrite ores. Kohle u. Erz, vol. 27, 1930, pp. 626-627.
Three percent copper ore was treated with ethyl, butyl, and amyl xanthates, with pine oil as a frother. With the same molar amounts of xanthate, the potassium butyl and potassium amyl xanthates were more efficient than potassium ethyl xanthate. By increasing the amount of the latter and prolonging the time, the potassium ethyl xanthate produced as high a recovery as the other two.

51. BAKER, J. A. A mill for the small gold mine. Min. and Met., vol. 13, 1932, pp. 209-216.
General article on the size and kind of treatment plant that may be justified for a small gold mine. Discusses relation between developed ore, grade of ore, mill recovery, and first cost of mill. Cites examples of small amalgamation and cyaniation mills and discusses methods of employing flotation in the treatment of gold ores.

52. BARSKY, GEORGE, and FALCONER, S. A. Differential wetting effects in flotation. Trans. Electrochem. Soc. (Prepr. 60), 1931, 10 pp.
A method, based on the work of Bartell on pigments, has been applied to determine the differential wetting effects of water and pine oil on the surface of mineral particles. The results show that reagents producing flotation alter the mineral so that pine oil displaces water at the surface. Depressants, on the other hand, have been shown to increase the water-wetting effect over that of pine oil. These results are consistent with the general theory of flotation.

53. BEAR, FIRMAN E., and ALLEN, LILBURN. Relation between fineness of limestone particles and their rates of solution. Ind. Eng. Chem., vol. 24, 1932, pp. 996-1001.

Pulverized limestones having different percentage contents of the various particle sizes were evaluated, in terms of some standard particles, as solution reduced this size with time. Tests showed that theoretical evaluations are nearly valid for finely pulverized separates but not for coarser separates, apparently as a result of diffusion.

54. BEATTY, G. H. The Randfontein Estates. Min. Mag. (London), March 1931, pp. 137-152.

Describes in detail mining and milling operations of Randfontein Estates Gold Mining Co. in Western Witwatersrand. The milling plant has a capacity of 220,000 tons of ore a month and uses crushers, stamps, tube mills, classifiers, corduroy tables, sand leaching plant, slime leaching plant, cyanide precipitation plant, and gold refinery. Flow sheets, details of equipment, metallurgical data, etc., are given.

55. BECKER, L. M. Sodium diethyl dithiophosphate (soda Aerofloat) as a flotation reagent. Tzvetnuie Metallui, 1931, pp. 984-989.

Both American and Russian soda Aerofloats gave unsatisfactory results in floating oxidized ores; on sulphide ores both gave very good results.

56. ____ Use of reaction products of phosphorus pentasulphide and cresol as flotation reagents. Tzvetnuie Metallui, 1931, pp. 477-486.

The purpose of the investigation was to produce Aerofloat, at present manufactured exclusively by an American company. The following preparation method was adopted: To 1 to 1.25 parts of finely ground phosphorus pentasulphide gradually add 3 parts cresol, and reflux 2 to 2.5 hours at 115 to 130°; toward the end of the reaction increase the temperature about 5 to 10° for 5 to 10 minutes. The reaction product is a dark green oil with strong odor of hydrogen sulphide. Reagents were prepared in this way from German cresol, Russian cresol, crude phenol and coal tar. Comparative flotation tests of Coounrade copper (oxide) ore and on Karsak-Pai ore (chalcopyrite and bornite) showed that the reagent prepared from Russian cresol equals American Aerofloat; the reagent prepared from German cresol was also satisfactory, but the phenol reagent was entirely unsatisfactory. The chief constituents of the reagent are phosphoric and thiophosphoric esters of cresol, thiocresols, excess cresol, and dissolved hydrogen sulphide. The experiments were performed entirely on a laboratory scale.

57. BEEMAN, NORVIL. The effect of hydrogen-ion concentration upon the sedimentation of clay. Jour. Am. Ceram. Soc., vol. 14, 1931, pp. 72-87.

The effect of hydrogen-ion concentration upon the rate of sedimentation was relatively great, which explains why changes in pH values produce changes in plasticity and viscosity. Elutriation is an uncertain process without pH control.

58. BEMIS, H. D. Milling methods and costs at the Pecos concentrator of the American Metal Co., Terarro, N. Mex. Inf. Circ. 6605, Bureau of Mines, 1932, 23 pp.
Describes a flotation plant of 600 tons capacity treating lead-zinc ore (replacement ore bodies in pre-Cambrian schists) containing pyrite, sphalerite, galena, chalcopyrite, marmatite, chalcocite, bornite, argentite, and proustite (named in order of abundance); the gangue minerals are mica, quartz, tourmaline, and hornblende. Flotation practice at this mill differs from the usual lead-zinc ore methods in that mica is removed from the ground pulp by flotation (using only cresylic acid as a reagent) before the flotation of lead. The mica concentrates are cleaned on tables to recover small quantities of sulphide minerals and discarded. The remainder of the flotation follows the usual two-stage lead and zinc treatment. No pyrite concentrates are made. Metallurgical data, flow sheets, costs, reagents, etc. are given, as well as a description of the equipment used.

59. BENEDICT, C. H. Milling at the Calumet & Hecla Consolidated Copper Co. Min. Cong. Jour., vol. 17, 1931, pp. 519-527.
Reviews the development of ore-dressing methods and outlines present practice and operating details. The Calumet (11-stamp), Hecla (17-stamp), Ahmeek (8-stamp), and Osceola (7-stamp) mills are described, as well as the Tamarack reclamation plant and the ammonia leaching plant. Typical flow sheets of milling operations on conglomerate and amygdaloid ore are given.

60. BERL, E., and SCHMIDT, A. A new screening device. Chem. Fabrik, 1932, p. 299.
Describes a shaking device which, when fitted with sieves, makes a good screening apparatus.

61. BERTHELOT, CH. Evolution and orientation of the coal industry. Washing and modern carbonization methods. Proc. World Eng. Cong., Tokyo, 1929, vol. 37, 1931, pp. 23-58.
Discusses coal washing and low-temperature and high-temperature carbonization.

62. _____ Modern methods of concentrating minerals. Rev. metal., vol. 27, 1930, pp. 535-543.
General review of flotation methods.

63. _____ New methods of purification of coal. Génie civil, vol. 100, 1932, pp. 6-10 and 29-35.
Pneumatic cleaning is inferior to wet washing, as far as ash content is concerned, but produces less moist product. Drawings and details of the equipment used in both processes are given.

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64. BIERBRAVER, ERNST, and GLEICHMANN, HUBERT. The recovery of a copper product from the siderite of the Eisenhardt mine and its concentration by flotation. Mitt. Kaiser Wilhelm-Inst. Eisenforsch. Dusseldorf, vol. 13, 1931, pp. 121-129.

The average composition of the ore was 31.40 percent iron, 5.7 percent manganese, 0.446 percent copper, and 18.70 percent insoluble. The copper was present as chalcopyrite. Other sulphides present were pyrite and arsenopyrite. The reagents used were phospho-cresol B (a commercial product) as collector and frother for the chalcopyrite and pyrite, sodium sulphite to depress the pyrite during flotation of the copper, sulphuric acid to reactivate the pyrite, high-molecular fatty acids to float the siderite, sodium hydroxide to make alkaline the siderite slime, and sodium silicate to deflocculate the gummy siderite froth. The steps taken were in the following order: (1) Conditioning period, with the addition of sodium sulphite to depress the pyrite; (2) frothing period, with flotation of the chalcopyrite with phospho-cresol; (3) flotation of the pyrite after acidifying with sulphuric acid; (4) conditioning period, during which the slime was made alkaline and waterglass added; (5) frothing period, with flotation of the siderite by high-molecular fatty acids. Of the total copper, 95.56 percent was recovered in 23.9-percent copper concentrates.

65. BIRD, B. M., and MARSHALL, S. M. Jigging, classification, tabling, and flotation tests on coals presenting difficult washing problems, with particular reference to coals from Pierce County, Wash. Bull. 337, Bureau of Mines, 1931, 132 pp.

With screen-sized feeds, classification was the most efficient cleaning method investigated, with jigging a close second. Slack sizes through a 3/16-inch Ton-Cap screen were cleaned most efficiently by a combination of classification, tabling, and flotation. An improved hindered-settling classifier was developed. High riffles markedly increased table efficiency. A new method of adjusting coal-washing tables was introduced. The analyses of beds 3, 7, 8, 10, and 11 from Pierce County are: Ash, 21.4 to 31.9 percent, average 25 percent; volatile (moisture- and ash-free), 25 to 32 percent, average 28.6 percent; sulphur (moisture- and ash-free), average 0.81 percent; and phosphorus, average 0.091 percent (after washing). The average coal had 46.1 percent between 1.35 and 1.70 specific gravity (bone). The specific-gravity distribution curves show the problem of preparing a washed coal of 12-percent ash is very difficult. Washing tests gave the following yields: 10 percent ash, 61.2; 11 percent ash, 66.8; 12 percent ash, 71.0; and 13 percent ash, 74.3 percent. Crushing below 3/8 inch has doubtful value.

66. BIRD, B. M., RICHARDSON, A. C., and COE, G. D. Washability studies of the Mary Lee bed at Hull mine, Dora, Ala. Rept. of Investigations 3067, Bureau of Mines, 1931, 24 pp.

The problem of washing the Mary Lee bed at the Hull mine is confined to ash reduction. A washed coal low enough in ash to permit its use in the manufacture of coke is not obtainable in the coarse sizes without recrushing to 3/16 inch.

67. BLACKBURN, W. H. Milling methods and costs at the lead-zinc concentrator of the Treadwell Yukon Co. (Ltd.) at Tybo, Nev. Inf. Circ. 6430, Bureau of Mines, 1931, 14 pp.
The Tybo mill is 70 miles northeast of Tonopah, Nev., has a capacity of about 320 tons a day and was placed in operation in May 1929. The plant uses straight flotation on an ore containing galena, sphalerite, and pyrite in a siliceous gangue. Lead concentrates and zinc concentrates are produced, the pyrite being depressed into the tailings. The ore is reduced to about 3/8 inch in a jaw crusher and a cone crusher, ground in single-stage conical ball mills in closed circuit with rake classifiers, and treated in Minerals Separation flotation roughers. Hearing-type flotation cleaners are used. Lead-circuit reagents are soda ash, sodium cyanide, zinc sulphate, ethyl xanthate, and cresylic acid. In the zinc circuit lime, copper sulphate, zanthate, and pine oil are added. Lead recovery in the lead concentrates is 71.3 percent, and zinc recovery in the zinc concentrates is 72.4 percent.

68. BLOCK, BERTHOLD. The removal of solid particles from liquids. Chem. App., vol. 17, 1930, pp. 244-246; vol. 18, 1931, pp. 17-18.
Considers settling, filtration, centrifuging, evaporating, drying, and variations of each of these unit processes theoretically and practically.

69. BOERICKE, W. F. Gold-milling developments in northern Ontario. Min. and Met., vol. 15, 1932, pp. 391-400
A detailed article, discussing flow sheets and general problems at the Kirkland Lake, Wright-Hargreaves, and Lake Shore mills in the Kirkland Lake district and the Hollinger, McIntyre, Dome, and Coniaurum mills in the Porcupine district. Flotation has been adopted in various combination with cyanidation at several mills, and the article gives typical flow sheets and comparative metallurgical data.

70. BOND, F. C. Calculating average particle size from screen analysis. Eng. and Min. Jour., vol. 132, 1931, p. 15.
Describes a method of calculating average particle size and gives table of micron values of different screens and three formulas for making calculations.

71. BORCHERDT, W. O. The Empire Zinc Co.'s operation at Gilman, Colo. Eng. and Min. Jour., vol. 132, 1931, pp. 251-261.
Detailed description of combined gravity and flotation concentrator at Gilman, 27 miles west of Leadville, Colo. The ore consists of pyrite, sphalerite, galena, siderite, calcite, dolomite, chalcopyrite, and other minerals. The deposit is partly oxidized, and the recovery of oxidized lead necessitated the combined treatment. The mill is underground, due to lack of space in the Eagle River Canyon. The mill uses jaw and cone crushers, rod mills and ball mills, jigs, tables, and flotation machines. Lead-circuit reagents are caustic soda, soda ash, sodium silicate, creosote, pine oil, and fuel oil. Zinc-circuit reagents are copper sulphate, soda ash, lime, sodium silicate, pine oil, and fuel oil.

72. BORNER, H. Application of a reclassification method to the flotation process for the improvement of ore-dressing results. Metall u. Erz, vol. 27, 1930, pp. 654-659.

Investigates distribution of metal in the flotation process with the aid of sieve and uniformity analyses of gold ores and dependence of metal recovery upon the grain size of the flotation slime. Whether this reclassification process can be applied depends upon the value of the metal being recovered.

73. BOUTARIC, A. Emulsions and their industrial application. Génie civil, vol. 97, 1930, pp. 467-468 and 488-489.

Foams are obtained with liquids of low surface tension in contact with air. In flotation processes the amount of oil needed is reduced greatly by bubbling of colloids. Near the point of reversal or change of type emulsions are particularly unstable. Thick emulsions can be destroyed by heating to 60 to 70°. To suppress a foam, destroy the stabilizing product.

74. BOWEN, MAX W. Flotation of gold ores offers new opportunities. Eng. and Min. Jour., vol. 133, 1932, pp. 448-449.

Discusses applicability of flotation in treating gold ores. Gold ores are divided into three classes and the use of flotation in the treatment of each class is discussed. Various combinations of flotation with other processes are described, and the types of flotation machines are considered. The functions of flotation reagents in gold circuits are discussed, and general factors to be considered in plant construction are given.

75. BRADLEY, JAMES. Mining and milling at the Spanish mine. Min. and Met., vol. 12, 1931, pp. 435-439.

The ore treated at the concentrator assays $3.00 to $6.00 in gold, 2.0 to 3.0 ounces silver, 0.6 percent copper, 1.5 percent lead, 2.0 percent zinc, 5.0 percent iron, and 20.0 percent barite. The sulphide minerals are chalcopyrite, galena, sphalerite, tetrahedrite, tennantite, bornite, and chalcocite. After crushing and grinding the ore is treated in Kraut flotation machines making lead concentrates, zinc concentrates, and iron concentrates and the flotation tailings are treated on tables making barite concentrates. Metallurgical results, flotation reagents, flow sheets, and other data are given.

76. BRADLEY, W. W. Renewed activity in California gold mining. Min. and Met., vol. 13, 1932, pp. 385-390.

After giving a short historical sketch of gold mining in California article describes briefly the application of flotation in the treatment of gold ores at the Kennedy, Idaho Maryland, Murchic, North Star, Empress, and Royal properties.

77. BRIGHTON, T. B., BURGENER, G., and GROSS, JOHN. Depression by cyanide in flotation circuits. Eng. and Min. Jour., vol. 133, 1932, pp. 276-278.
Studies effect of cyanides on floatability of pure minerals. The abstraction of cyanide from solution varies with different minerals but increases with increase in mineral surface. The degree of depression of a mineral increases with increased cyanide abstraction. The solubilities of metal xanthates in cyanide solutions are given. Copper xanthate and zinc xanthate are soluble, but lead xanthate insoluble in cyanide. Cyanides render minerals nonfloatable by removing xanthate films. With zinc sulphate zinc cyanide may be precipitated on the mineral surface. Cyanide forms ferrocyanides and ferricyanides with pyrite which may be adsorbed. Other conclusions are given.

78. BRING, GUST. G. Concentration of Swedish iron ores. Jernkontorets Ann., vol. 114, 1930, pp. 64-132.
Results of tests of Swedish magnetite and hematite, using gravity concentration, flotation, and magnetic separation.

79. ____ Investigations on magnetic separators. Jernkontorets Ann., vol. 116, 1932, pp. 255-278.
Discussion of magnetic field strengths used on high- and low-magnetic ores.

80. BRINSMADE, R. B. Mining industry of Santa Eulalia, Mexico. Arizona Min. Jour., vol. 15, 1931, pp. 3-4.
The article (third of a series) describes operations at the customs flotation plant of Industrial Co. (Potosi Mining Co.) treating lead-zinc-iron ore from the Potosi and Calera mines. The crushed ore is ground in rod mills and floated in Minerals Separation and Forrester machines making three concentrates. Reagents used are sulphurous acid, xanthate, cresylic acid, copper sulphate, and creosote. Flow sheets and metallurgical data are given.

81. BROWNLIE, DAVID. The Lessing coal-cleaning process. Combustion, vol. 1, no. 10, 1930, pp. 42-46.
Describes process and gives operating results.

82. BUCHELLA, F. H. The primary crushing plant at McGill, Nev. Arizona Min. Jour., vol. 16, 1932, pp. 5-7.
A detailed description of the new crushing equipment of Nevada Consolidated Copper Co. at McGill, Nev. The ore cars are dumped by a Wellman machine onto a grizzly to an 84- by 66-inch jaw crusher designed to handle the maximum blocks of ore that can be passed by the 4-yard pit shovels. The crusher product goes to Symons cone crushers.

83. BURCHOLD, C. Selective flotation of lead-tin ores in Mexico. Metall u. Erz, vol. 28, 1931, pp. 541-545.
Describes Santa Eulalia installation of the American Smelting & Refining Co. The finely ground ore is treated with sodium sulphide. The nonsulphidized tin and magnetic iron are removed, and dried iron-tin concentrate is separated electromagnetically.

84. **BURGESS, B. C.** Methods and costs of milling feldspar at the Minpro plant, Tennessee Mineral Products Corporation, Spruce Pine, N. C. Inf. Circ. 6488, Bureau of Mines, 1931, 22 pp.

Describes in detail milling operations at a plant having a capacity of 4,000 tons of ground spar a month. The ore is received from as many as 60 mines near by and reduced to 4 inches in a jaw crusher, the minus ½-inch portion removed on a grizzly and discarded, and the remainder carefully hand-sorted in two grades each of soda spar and potash spar. Flint and mica also are removed. The hand-sorted products are dried, conveyed to mixing bins, ground in conical mills in closed circuit with air separators, and stored for shipment. More than 90 grades of soda-potash mixtures are prepared and marketed.

85. **CALLOW, J. M.** The Mount Isa enterprise. Eng. and Min. Jour., vol. 133, 1932, pp. 371-375.

Describes mining, milling, and smelting at Mount Isa, Queensland, Australia. The ore consists of lead carbonate and lead sulphide, with some zinc sulphide. Treatment is by a combination of table and flotation concentration.

86. **CAMPBELL, A. B., HOWES, WARREN, and ODE, W. H.** Salts in tri-State mill waters: Their ill effect on the flotation of blende and their removal. Rept. of Investigations 3149, Bureau of Mines, 1932, 24 pp.

The use of acid mine waters for milling in the tri-State zinc district of Missouri, Kansas, and Oklahoma has caused corrosion of mill equipment and difficulties in flotation. The acid waters were most abundant in the eastern part of the Picher district, where carbonates were absent in the ore bodies. If carbonates, chiefly calcite, were found in the tailings piles the waters did not contain enough acid or metallic salts to cause trouble. Analyses of the troublesome waters sometimes showed more than 5,000 parts per million of free acid and of dissolved salts in the form of sulphates. The principal corrosive agents are free acid and ferric sulphate. The principal flotation toxin is ferrous sulphate; zinc sulphate requires minor consideration. Appreciable quantities of the sulphates of calcium and magnesium caused little or no trouble. The most satisfactory treatment is the selective removal of ferrous and ferric iron and free acid by alternate periods of aeration and neutralization. The aeration oxidizes the ferrous iron; the neutralization removes free acid and the ferric iron. Neutralization may be accomplished most cheaply by upward percolation through high-grade limestone; low-grade limestones are ineffective. Bases, such as lime and caustic soda, act more quickly but cost more.

87. **CAMPBELL, THOMAS P.** Examples of hydrometallurgical and electrometallurgical operations. Part II. Gold and silver. Colorado Sch. Mines Mag., vol. 21, 1931, pp. 9-13.

Discusses various phases of percolation, the arrangement of plant, slime or agitation leach, precipitation, refining, regeneration of cyanide and costs. Percolation is divided into two classes, dry and wet treatment. The former is applied to materials from tailing ponds and dry-crushing mills and the latter to ores delivered wet as a flowing pulp from mills or previous treatment. A flow sheet is shown for a 500-ton combination flotation and cyanide plant. Direct operating costs and metallurgical results of Northern Ontario mills are tabulated.

88. CAMPBELL, THOMAS P. Part III, Refining of gold and silver. Colorado Sch. Mines Mag., vol. 21, 1931, pp. 11-16.
In acid parting, hot, concentrated sulphuric acid is replacing nitric acid. Acid parting may be outlined as: (1) Blending of the bullion and preparation for parting, (2) solution of silver and base metals, (3) treatment of residues for gold, and (4) treatment of the solution for silver. The Moebius and Balbach refining processes for silver and the Wohlwill process for gold are described.

89. CANADIAN DEPARTMENT OF MINES AND METALLURGY. Some problems in the treatment of gold ores. Bull. 235, 1931, pp. 1250-1261.
Problems in treating five classes of gold ores are considered. These classes are (1) free milling, (2) free gold and pyrite, (3) containing copper, (4) arsenical and (5) containing interfering gangues, such as graphitic material, soluble salts, etc.

90. CANADIAN MINING JOURNAL. The International Nickel Co. of Canada (Ltd.). Vol. 51, 1930, pp. 1034-1035.
General description of the mines and plants of the International Nickel Co., including an article on the Inco concentrator by W. T. Mac Donald.

91. CARMAN, J. B. Milling methods at the Questa concentrator of the Molybdenum Corporation of America, Questa, N. Mex. Inf. Circ. 6551, Bureau of Mines, 1932, 14 pp.
The ore treated in this 50-ton flotation plant contains about 5 percent molybdenum in the form of molybdenite. Other minerals are quartz, pyrite, feldspar, biotite, fluorite, rhodochrosite, and calcite, enclosed in granodiorite country rock. Operations were started in 1923. The mine ore is reduced to 1½ inches in a jaw crusher and ground to about 70 percent minus 200-mesh in a ball mill-classifier unit. The pulp is treated in Callow pneumatic machines, cleaned twice, dried, and sacked for shipment. Concentrates contain about 71 percent molybdenum sulphide, and the recovery is 86.8 percent. Flotation reagents are sodium cyanide and two kinds of pine oils. The report reviews milling operations, describes equipment used, and includes metallurgical and cost data and a flow sheet.

92. CASTIGLIONI, ANGELO. Ammonium hydroxide as a deflocculant for clay suspensions. Industria Chimica, vol. 5, 1930, pp. 1465-1468.
Ammonium hydroxide (specific gravity, 0.888), when added to clay suspensions in proportions as low as 0.02 percent by weight increases the fluidity markedly, making possible the use of much more concentrated suspensions. In the manufacture of molded ceramic ware it permits the use of suspensions of a density of 1.85, which remain as fluid as suspensions of 1.72 when sodium silicate is used as the deflocculant.

93. CHINKIN, N. N. Dressing of iron ores in the Central Industrial Region. Inst. Mekh. Obrabotki, Pol. Isk., 1931, 37 pp.
Describes log-washing and magnetic separation tests and plants treating brown iron ore in Russia.

94. CLARK, ALLAN J. Milling, amalgamation, cyaniding. Eng. and Min. Jour., vol. 132, 1931, pp. 298-304.
The development of the present milling methods at Homestake is outlined. The process used is amalgamation, followed by cyanidation of the classified sands and slimes. Minor sulphides occur in the ore; but the bulk of the gold is carried in nonsulphide minerals, precluding any possibility of concentration followed by cyanidation of the concentrates. Fine grinding is required for complete liberation of the gold, and amalgamation is necessary to reduce time of cyanidation. Separate cyanide treatment of sands and slimes is more economical than all-slime cyanidation. Rod mills are more satisfactory than ball mills, and gravity stamps are still in use. Flow sheets of the various plants are given.

95. _____ Milling methods and costs at the Homestake mine, Lead, S. Dak. Inf. Circ. 6408, Bureau of Mines, 1931, 25 pp.
Describes in detail crushing, stamp milling, amalgamation, and cyanidation at various plants of the Homestake Mining Co. This company operates two crushing plants, 3 stamp mills, 2 sand-leaching plants, and 1 slime-leaching plant. The report gives a bibliography of references to earlier articles on geology, mining, and milling at Homestake. Flow sheets and metallurgical data and a milling-cost summary are included. The ore bodies occur in altered dolomitic limestone containing cummingtonite and chlorite, with minor quantities of pyrite, arsenopyrite, and pyrrhotite. The ore carries about $5.00 in gold a ton. Amalgamation recovers 62.8 percent of the gold, cyanidation of sands 20.6 percent, and cyanidation of slimes 9.6 percent, a total recovery of 93 percent.

96. CLARKE, W. G., and MOORE, B. H. Report on an investigation into the possibility of effecting a separation of cassiterite and tantalite from cassiterite-tantalite concentrates from Greenbushes. Sch. Mines Western Australia, Bull. 6, 1931, pp. 32-34.
Mechanical methods of separation have been found unsatisfactory. The most successful chemical method involves the following stages: (1) Reduction of cassiterite by heating the concentrate in a current of reducing gas; (2) conversion of the reduced tin to tin chloride by the action of dry chlorine at controlled temperatures; (3) volatilization of tin chloride by heating and recovery in liquid form by condensation; and (4) reconcentration of the residues by gravity methods to produce a high-grade tantalum-columbium concentrate by eliminating hematite and other gangue minerals.

97. CLARKE, W. G., and MOORE, B. H. Report on an investigation into the roasting and cyanidation of flotation concentrates from Wiluna Gold Mines (Ltd.). Sch. Mines Western Australia, Bull. 6, 1931, pp. 19-31.

This investigation was conducted to determine: (1) The most suitable temperature of roasting, especially with regard to the starting and finishing heat; (2) the advisability of washing with water or other solutions before cyaniding; and (3) the consumption of chemicals for cyaniding. Three stages in roasting of concentrates are: (1) Sublimation of arsenious oxide and partial oxidation of pyrite (340 to 400°); (2) further oxidation of pyrite (420 to 500°) and ignition of the sulphur at 450°, causing a temperature rise to 500° without external heat; and (3) swelling of the charge (500 to 600°), probably due to liberation of carbon dioxide. Temperatures should be maintained at each stage until changes are completed to avoid the formation of arsenates and sintering and serious volatilization of gold due to excessive heating. The water washing, grinding, and cyanidizing of the calcine are discussed fully and results tabulated. Up to 6 percent gold is lost during the second stage of the roast, possibly due to the presence of chlorine in the concentrate. A 0.08-percent free cyanide solution quickly extracts all gold amenable to cyanide extraction. The consumption of chemicals during cyanidation of the calcine decreases as the finishing temperature of the roast is increased and with a finishing temperature of about 800°, should not exceed 0.5 pound (227 grams) cyanide and 2.27 kilograms lime per ton of calcine. A 93-percent extraction of gold is equivalent to 88 percent of the gold in the raw concentrate.

98. _____ Report on an investigation into the treatment of cupriferous gold tailings from Gabanintha, Murchinson Goldfield. Sch. Mines Western Australia, Bull. 6, 1931, pp. 3-14.

Twenty-two tables are shown covering studies on roasting, amalgamation, grinding, flotation, and cyanidation. The work may be summarized as follows: (1) Fine grinding is essential for successful amalgamation and cyanidation; (2) amalgamation during fine grinding in pans of a composite sample from these dumps can yield an extraction of over $2.50 in gold per ton; (3) flotation or strake concentration yields unsatisfactory extraction of both gold and copper; (4) cyanidation of raw ore after fine grinding and amalgamation yields a further extraction of $1.75 per ton without excessive consumption of cyanide but requires a vacuum filtration plant; (5) treatment with sulphuric acid to extract copper before and after gold treatment is impracticable; (6) the most satisfactory treatment consists of roasting, amalgamation during grinding in pans, and cyanidation of the roasted product by agitation or percolation. This method yields 92 percent of the gold, with a reasonable consumption of cyanide and lime and without excessive solution of copper.

99. _____ Report on the efficiency of Western Australia eucalyptus oils as frothing agents in the flotation of ores. Sch. Mines Western Australia, Bull. 6, 1931, pp. 35-40.

The best method of determining the efficiency of any particular oil is to make a series of comparative flotation tests. Four oils containing no phellandrene and a high percentage of cineole and one containing a large quantity of phellandrene and no cineole were used. Three Western Australia oils give results comparing favorably with two standard oils.

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100. CLARKE, W. G., and MOORE, B. H. Report on the investigation into the treatment of clean-up material from old plant, Lake View and Star (Ltd.) (Gold metallurgy). Sch. Mines Western Australia, Bull. 6, 1931, pp. 15-19.

The clean-up materials investigated comprised: (1) Residue from amalgamation barrel treatment of slag, (2) roaster flue dust, and (3) material from surface clean-up at the Golden Horseshoe plant. Results of the cyanidation of these materials are tabulated and indicate that further investigation of amalgamation barrel residue is warranted.

101. ____ Report on the investigation into the treatment of sulphide ore from the Patricia leases of the Kimberley Oils Options Co., at Edjudina. Sch. Mines Western Australia, vol. 6, 1931, pp. 40-45.

An investigation to determine the most suitable method of treatment of the sulphide ore from the Patricia leases is outlined. The following conclusions are drawn: (1) Efficient amalgamation during fine grinding is capable of yielding a gold extraction of 90 percent; (2) on account of the ready decomposition of the chalcopyrite, this ore is not amenable to cyanidation; (3) the bulk of the copper and a large proportion of the gold not extracted by amalgamation can be extracted by suitable methods of concentration.

102. CLEVENGER, GALEN H. The development of the cyanide process. Trans. Electrochem. Soc., Prepr. 60, 1931, 9 pp.

Presents short historical review, leading up to the commercial successes of McArthur and Forrest. The efficiency of potassium cyanide compared with sodium cyanide and the various processes for making cyanides are considered. To-day most of the cyanides used in cyanidation are derived from cyanamide. As to the future of cyanidation, no better process has been found for gold and silver. Flotation will not displace it. There is a distinct field, namely, the cyanidation of base metals, which holds forth promise and ought to be further investigated.

103. COGHILL, WILL H., and CLEMMER, J. BRUCE. Flotation of the nonsulphides. Eng. and Min. Jour., vol. 133, 1932, pp. 136-137.

Limestone, phosphate rock, bauxite, fluorspar, rhodochrosite, manganese oxides, barite, siderite, chromite, scheelite, ferberite, cyanite, beryl, and spodumene are considered. Some have been worked successfully on a commercial scale and the remainder on a laboratory scale. Successful flotation of such ores requires proper selection and proportions of pine oil, sodium oleate, oleic acid, sodium carbonate, sodium hydroxide, sodium silicate, etc. Some ores require sulphuric acid in the cleaning treatment.

Original from THE OHIO STATE UNIVERSITY

104. COGHILL, WILL H., and CLEMMER, J. BRUCE. Soap flotation of the nonsulphides (limestone, phosphate rock, bauxite, fluorspar, rhodochrosite, manganese oxides, barite, siderite, chromite, scheelite and ferberite, and cyanite). Am. Inst. Min. and Met. Eng. Tech. Pub. 445, 1932, pp. 3-18.

In the soap flotation of nonsulphide ores the first objectives are the selective flocculation of the mineral and the dispersion of the gangue, as in other flotation processes. The chief role of the soap in the flotation described here is its selective absorption. The "suds" it makes when used alone are too tough and too stable to be satisfactory, and additions are necessary. From 100 to 150 parts of soap in a million parts of water are usually enough to flocculate both the mineral and to some extent the gangue, and this latter effect must be minimized by using alkaline dispersing agents, such as sodium silicate or sodium carbonate. About 5 parts of pine oil per million of water is added to impart brittleness to the froth. The pine oil probably affects the degree of flocculation as well. In view of these additions, the authors point out that the work described here is not the same as that by Adams and others (Eng. and Min. Jour., vol. 132, 1931, pp. 54-55), which is more strictly "soap flotation". Sodium silicate is an important reagent in nonsulphide flotation; the orthosilicates and metasilicates are the two forms preferred. Bicarbonates are undesirable alkaline dispersants, and carbon dioxide in general is regarded with suspicion. An elevated temperature and the use of calcium oleate as a flotation agent are of value for hard waters. Sulphuric acid has been useful for cleaning purposes. A positive means for air control is important. Machines using a mechanical beating seem better than air machines. Comments are made on various tests on the minerals listed, and some bibliographical references are given.

105. COGHILL, WILL H., and O'MEARA, R. G. Milling methods and costs at a Flat River (Mo.) mill. Inf. Circ. 6658, Bureau of Mines, 1932, 35 pp.

Describes in detail treatment by combined gravity and flotation concentration of ore containing galena in a dolomitic gangue, with minor amounts of silver, copper, iron, and zinc as sulphides. Ore is dry-crushed to 3-mesh, ground in rod mills, and sized by hydraulic classification; the sands are tabled, and the slimes are thickened and treated by flotation. The report gives a brief history of the milling operations, which started in 1906 as a straight jig and table plant, through the adoption of flotation in 1914, the discard of jigs in 1927, and the adoption of hydraulic classification in 1928 to its present form, treating 5,000 tons a day. Describes in detail machines used and metallurgic results; includes milling-cost data.

106. COGHILL, W. H., HOWES, WARREN, and COOKE, S. B. Mineragraphic aid in the concentration of manganiferous iron ores. Eng. and Min. Jour., vol. 131, 1931, pp. 361-365.
Two manganiferous ores, representative of large deposits in the Cuyuna district, Minnesota, were studied by table concentration, flotation, roasting and magnetic separation, and mineragraphic methods. The ores were quite similar in composition and appearance but differed widely in results of separation by the methods mentioned. A microscopic study led to the conclusion that ore A did not give good flotation concentrates because of the presence of carbonates intimately associated with siliceous gangue. Ore A responded to table concentration, because the greater portion of the manganese and iron minerals was fairly well liberated from the gangue at table size. Ore B could not be tabled satisfactorily because of intimate locking of the ore minerals with the quartz gangue. Magnetic separation was impossible, because the manganese and iron minerals were interlocked closely. Many photomicrographs and camera-lucida drawings are shown.

107. COLE, DAVID. Crushing practice in the Southwest. Min. and Met., vol. 12, 1931, pp. 76-80.
Discusses in a general way development and improvement of crushing machines and flow sheets in the Southwest, particularly at the large copper concentrators in Arizona. Especial attention is given the development of the Symons cone crusher and the various ways in which it can be used in a crushing circuit. Some interesting comparisons with previous practice are drawn; in all cases the Symons cone crusher effected greater reduction, larger capacity, and lower costs.

108. COLE, GEO. E. Progress in metal mining in Manitoba. Canadian Inst. Min. and Met., Bull. 238, 1932, pp. 39-57.
Describes flotation, cyanidation, leaching, and copper smelting at Flin Flon and milling methods at Sherritt-Gordon.

109. CROWFOOT, ARTHUR. Concentrating methods and costs at the Morenci concentrator of the Phelps Dodge Corporation, Morenci, Greenlee County, Ariz. Inf. Circ. 6460, Bureau of Mines, 1931, 36 pp.
Reviews in detail past and present milling operations at a 5,000-ton concentrator, using a combination of table and flotation concentration in the treatment of a copper ore containing chalcocite and pyrite associated with monzonite-porphyry gangue minerals. The original mill, placed in operation in 1906, had a capacity of 900 tons a day and used jigs, tables, and vanners. Flotation of gravity tailings was adopted in 1916, and all vanners and jigs have since been discarded, as the present mill uses tables and flotation. Tables are retained in the present flow sheet largely because of smelter requirements for pyritic concentrates. Under present operations mine ore is reduced to 6-mesh by a primary jaw crusher, 2 cone crushers, and 4 sets of rolls. This material is concentrated on tables, final concentrates are removed, and the table tailings are ground to 48-mesh in 6 ball mills and rake classifiers. The pulp passes to the primary flotation machines, making rougher concentrates, which are cleaned once, and tailings, which pass to 5 bowl classifiers in closed circuit with 5 secondary ball mills grinding to 65-mesh. Secondary flotation machines remove concentrates and leave final tailings. Flotation reagents used are lime, pine oil, and xanthate.

110. CURWEN, H. C. The treatment of gold ores by flotation. South African Min. and Eng. Jour., vol. 43, part I, 1932, pp. 339-340, 441, and 487-488.

Gold and silver ores are classified into seven major groups for which five treatment schemes are proposed. The advantages of a flotation-cyanide plant over an all-cyanide plant are given. Experimental results and operating practice at McIntyre Porcupine mill, Independence mill, and the Alaska Juneau mill are used for illustration.

111. DALTON, M. P. Milling methods and costs at the Morning concentrator of the Federal Mining and Smelting Co., Mullan, Idaho. Inf. Circ. 6587, Bureau of Mines, 1932, 11 pp.

The ore treated at the Morning mill consists of sulphides of lead and zinc, with quartz and siderite. The original mill used jigs, and tables and flotation equipment (MacQuiston tubes) were added in 1911. The mill was converted into a straight flotation plant in 1936. Under present milling methods the ore is reduced to 1 inch in jaw and cone crushers and ground in ball mills in two stages. Lead flotation is done in O. & D. machines, using zinc sulphate, sodium carbonate, Minerac "A", and Barrett no. 4. Lead middlings and cleaner tailings are reground and returned to the rougher. In the zinc section, Minerals Separation roughers and Callow cleaners are used, and the reagents are copper sulphate, xanthate, and Barrett no. 4. Zinc middlings also are reground. The report gives metallurgical and cost data, flow sheets, and descriptions of equipment used.

112. DAMON, A. C. Insuring mining investments. Arizona Min. Jour., vol. 15, 1931, pp. 6-7.

A short article explaining briefly the economic and metallurgic advantages of small portable-type test mills. The author stresses the economic phases, such as the low initial cost, ease of moving, and high salvage value compared with a small permanent mill. Metallurgic advantages include standard equipment and a wide flexability in flow sheets. Several types of flow sheets are indicated for precious or base-metal ores.

113. DANNENBERG, FRITZ. The coagulation of quartz and bolus suspensions by gelatin sols, carragheen sols, and electrolytes. Kolloidchem. Beihefte, vol. 31, 1930, pp. 447-468.

Suspensions of electrolyte-free quartz and bolus are prepared carefully. To these suspensions are added dilute gelatin and carragheen sols. Gelatin up to concentrations of 0.01 gram per liter renders the suspensions more sensitive to precipitation alone and in the presence of electrolytes. The ions of potassium, barium, and aluminum exhibited approximately the effect expected from the Schutz-Hardy rule. The gelatin is considered adsorbed on the surface of the suspended particles in the form of a hydrophobic colloid. When higher concentrations of gelatin are used much higher concentrations of electrolyte are needed for flocculation, indicating that in such concentrations the gelatin is adsorbed as a hydrophilic colloid. Carragheen sols do not influence coagulation of the suspensions. The Schutz-Hardy rule does not hold for addition of electrolytes to suspensions when carragheen sols are present. The time of aggregation of the bolus suspensions is determined for increasing amounts of gelatin in the presence of 0.01 mole potassium chloride solution. Several minerals exhibit behavior similar to that of the quartz and clay suspensions.

114. DAVIS, A. C. Coal grinding. Cement and Cement Manuf., vol. 3, 1930, pp. 1050-1072.

Describes the coal-grinding equipment used in the cement industry, with particular reference to unit systems.

115. _____ Crushing and preliminary grinding in portland cement manufacture. Cement and Cement Manuf., vol. 3, 1930, pp. 813-822.

Describes jaw, gyratory, roll, hammer, and disk crushers and ball mills. Gives power requirements and characteristic size-reduction ratios are given for the various mills.

116. _____ Grinding of cement clinker. Cement and Cement Manuf., vol. 3, 1930, pp. 1184-1195.

Discusses jaw and cone crushers and ball and tube mills.

117. DAVIS, NOAH S., JR. Preferential wetting of solids by liquids. Ind. Eng. Chem., vol. 24, 1932, pp. 1137-1140.

Presents results of a study of the preferential wetting of water versus xylene, toluene, aniline, turpentine, chloroform, nitrobenzene, amyl alcohol, cottonseed oil, linseed oil, paraffin oil, mineral oil, lard oil, rosin oil, olive oil, oleic acid, corn oil, oil of sesame, sperm oil, castor oil, coconut oil, furfural, tung oil, and other substances for sulphur, precipitated calcium carbonate, zinc oxide, lead carbonate, sphalerite, pyrolusite, hematite, malachite, galena, smithsonite, covellite, realgar, cerussite, stibnite, and pyrite.

118. DEAN, R. S. Bibliography of the metallurgical work of the U. S. Bureau of Mines in 1930. Inf. Circ. 6449, Bureau of Mines, 1931, 7 pp.

Over 60 references are given to articles on fundamentals, ore dressing, iron and steel, hydrometallurgy, manganese investigations, rare and precious metals, nonferrous metals, laboratory apparatus, methods, and discussions.

119. DEAN, R. S., and GROSS, JOHN. Explosive shattering of minerals as a substitute for crushing preparatory to ore dressing. Rept. of Investigations 3118, Bureau of Mines, 1932, 5 pp.

Preliminary report on the laboratory investigation of the shattering effect on mineral grains caused by the sudden release of pressure on superheated steam. Quartz and galena particles were soaked in water and heated to various pressures in a sealed container. The sudden release of the pressure not only ruptured the grains but indicated a preferential shattering effect.

120. DERINGER, D. C., JR. Correlating metal prices with concentration practice. Min. and Met., vol. 12, 1931, pp. 443-445.

Highest economic recovery is not necessarily attained with highest metallurgic recovery. Best economic recovery is determined by transportation and smelting charges and by sales prices of metals. Raising the grade of the concentrates 1 percent frequently results in higher economic recovery, even if losses of tailings are much higher. This is increasingly true as metal prices decline. In this article the author describes methods and formulas for setting up a series of curves from which the most economical ratios of grade of ore, grade of concentrates, grade of tailings, price of metals, etc. can be determined quickly, thus eliminating the laborious calculations necessary on each individual determination.

121. DE VANEY, F. D., and COGHILL, WILL H. Concentration tests on tailings from the washing plants of the Mesabi range, Minnesota. Rept. of Investigations 3052, Bureau of Mines, 1930, 23 pp.

The current tailings from western Mesabi washing plants contain about 20 percent iron, but through natural concentration by wind and water the bulk of the tailings in the sludge ponds now contains 45 to 50 percent iron. Roasting the sludge-pond tailings, followed by magnetic concentration, yields a concentrate containing about 64 percent iron, 0.042 percent phosphorus, and 8 percent silica with iron recovery of 97 percent. The tenor of the nonmagnetic tailing is about 5 percent iron. By classification and tabling concentrates were produced assaying 61 percent iron, 0.0343 percent phosphorus, and 8.85 percent silica, with iron recovery of 82 percent. Tailings contain about 30 percent iron.

122. _____ Concentration of the rake discharge from a bowl classifier in a washing plant of the Mesabi range, Minnesota. Rept. of Investigations 3148, Bureau of Mines, 1932, 7 pp.

Fines from ores of the jigging type are readily concentratable by classification and tabling. Bowl-rake products containing 14 to 20 percent silica are even more amenable to treatment. From such material a large percentage of the iron can be recovered direct from the first spigots of a classifier; only the later spigots require tabling. From rake products containing 20 percent silica, approximately 45 percent would be recoverable as classifier concentrates; all the overflow and 5 percent can be discarded, leaving 50 percent to be tabled. From a table feed assaying 44.40 percent iron, 34.50 percent insoluble, and 33.68 percent silica concentrates were produced assaying 62.98 percent iron, 9.00 percent insoluble, and 8.00 percent silica, and a tailing of 21.32 percent iron, 67.42 percent insoluble, and 65.15 percent silica. The concentrate carried 79.78 percent of the total iron and the tailing 84.17 percent of the insoluble.

123. DE VANEY, F. D., and COOKE, S. R. B. Laboratory concentration of the Missouri iron ores of Iron Mountain and Pilot Knob. Missouri School Mines and Met. Tech. Ser. 11, no. 3, 1930, 38 pp.

Iron Mountain ore is a complex mixture of hematite, garnet, quartz, calcite, and other minerals. Because of the high specific gravity of the garnet, gravity concentration of this ore is difficult and yields, at best, only medium-grade concentrates. Magnetic concentration of the roasted ore gives high recoveries and high-grade concentrates provided the ore treated is finer than 28-mesh. For satisfactory roasting, the ore must be crushed to a fine size and roasted at 600° or more. Pilot Knob ore is essentially an intimate mixture of hematite and quartz. It does not concentrate readily, and apparently the only means of producing a satisfactory product is magnetic concentration of the roasted ore, which requires grinding to 150-mesh or finer to free the silica.

124. DEWET, J. P. The new San Antonio mill. Canadian Min. Jour., vol. 52, 1931, pp. 812-813.

Brief description of the mine and new 150-ton cyanidation plant at San Antonio Gold Mines (Ltd.) on Rice Lake near Gabrielle, southeastern Manitoba. Flow sheet is given.

125. DICKSON, R. H., and SMITH, E. M. Milling methods and costs at the Verde Central concentrator, Jerome, Ariz. Inf. Circ. 6489, Bureau of Mines, 1931, 12 pp.

The Verde Central mill is a 400-ton straight flotation plant built in 1928; it treats copper ore containing chalcopyrite, pyrite, and quartz. The ore is reduced to 1/4 inch in jaw, cone, and roll crushers and ground in two conical ball mills in closed circuit with rake classifiers. The ground pulp is floated in two primary and two secondary MacIntosh machines making rougher concentrates and final tailings. Concentrates are cleaned in a fifth MacIntosh machine. Ore treated in 1930 averaged 2.67 percent copper. Ratio of concentration was 7.95 into 1; concentrates averaged 19.83 percent copper; recovery was 93.3 percent. Total milling costs were $1.26 a ton. Flotation reagents used are lime, xanthate, and pine oil.

126. DIXON, JOHN. Milling practice of the Kirkland Lake Gold Mines (Ltd.), Kirkland Lake, Ontario. Inf. Circ. 6508, Bureau of Mines, 1931, 13 pp.

A description of milling methods and costs at a 200-ton all-slime straight cyanidation mill at Kirkland Lake, Ontario. The plant employs two-stage grinding in ball and tube mills, agitation, countercurrent decantation, and filtration. A Crowe-Merrill plant is used for precipitation. The ore treated contains about $13.00 in gold in altered and silicified lamprophyre, quartz porphyry, syenite, and diabase. Total milling costs are $1.389 a ton.

127. DUNCOMBE, CHARLES G. The Kelly tube and the sedimentation of portland cement. Jour. Phys. Chem., vol. 36, 1932, pp. 31-51.

A new formula is proposed that takes account of leakage to the larger tube, neglected by Kelly. Calculation of the percentage settled directly from the readings simplified experimental work. Evaporation from the side arm of the tube introduces error that is reduced but not eliminated by Kelly's modification of his original tube. With cement containing coarse particles that settle rapidly the meniscus in the side tube lags behind the rate of settling. The error is not obviated easily by using viscous suspending liquids because of trouble with drainage in the main tube. A modified Wiegner apparatus is described that eliminated evaporation, has almost no lag, permits the use of viscous liquids, and is easy to manipulate.

128. DUNSTAN, E. T. The extraction of copper from oxidized ores by cyanide solution. Jour. Chem. Met. and Min. Soc. South Africa, vol. 31, 1931, pp. 190-194.

Cyanide extraction of copper from a malachite ore from Katanga containing 9.36 percent copper resulted in high cyanide consumption. Electrolytic regeneration was impossible. Details are given.

129. ELLINGSON, FLOYD C. Milling test of an oxidized gold ore from the vicinity of Rockford, South Dakota. Black Hills Eng., vol. 19, 1931, pp. 121-129.

Amalgamation tests showed 50 percent of the gold recoverable by amalgamation in bottles, with prospects of larger yields by using a muller. Cyaniding on 100-mesh ore gave 96 percent recovery. The ore does not produce enough slime to interfere with percolation. Cyanide consumption was about 0.6 pound per ton of ore; lime consumption was 20 pounds.

130. ELLIS, E. E. Universal Exploration Co. operations in East Tennessee. Min. Cong. Jour., vol. 17, 1931, pp. 4-7.

Describes in detail mining and milling at Jefferson City, Tenn. The new mill treats zinc sulphide ore by flotation, and older mills treat zinc carbonate ore by jigging. Flow sheets, equipment used, flotation reagents, etc., are noted.

131. ELLIS, E. T. Fine chemicals as precious-metal precipitants. Metal Ind. (London), vol. 38, 1931, pp. 377-378.

Chemicals that may be employed to precipitate gold are: Ammonium salts, such as oxalate and sulphide, antimony trichloride, ferrous sulphate, magnesium oxide, potassium salts, sodium oxalate, stannous chloride, and zinc oxide. Other precipitating agents for gold and some of the other precious metals are formic acid, gallic acid, mercurous nitrate, oxalic acid, acetic acid, potassium citrate, potassium sulphocyanate, potassium tartrate, sodium sulphide, sodium tartrate, tannic acid, and tartaric acid. In all instances the chemicals should be of high purity.

132. EMMONS, R. C. A note on gravity separation. Am. Mineralogist, vol. 15, 1930, p. 536.

The unsatisfactory nature of gravity separation by means of heavy solutions on finely divided material is accounted for best by adhesion, both of grain to grain and grain to air bubble. This can be eliminated by strong evacuation of the separating vessel. A few minutes is sufficient. Rapid stratification results when the liquid is allowed to stand, not necessarily under vacuum.

133. ENGINEERING AND MINING JOURNAL. Alaska Juneau milling system. Vol. 131, 1931, p. 369.

Quantitative flow sheet of Alaska Juneau mill, showing tons of material at each step in the operation.

134. _____ Current flotation practice at Broken Hill. Vol. 131, 1931, p. 552.

Briefly reviews concentration operations (historical sketch) at four companies (Zinc Corporation, Broken Hill South, North Broken Hill, and Sulphide Corporation) operating in the Broken Hill field. North Broken Hill is the only mill that uses straight flotation concentration. The other mills are combined gravity and flotation. Zinc Corporation and Broken Hill South use alkaline circuits. Two tables are given, one comparing metallurgical data and the other the flotation reagents used.

135. _____ Gold metallurgy in Western Australia. Vol. 133, 1932, pp. 10-12.

A general article on the mills in the Kalgoorlie section. Describes new process and equipment at Lake View and Star and Wiluna mills. Mentions the Boulder Perseverance and Sons of Gwalia plants.

136. _____ Gravity milling unit developed in California. Vol. 133, 1932, p. 229.

Describes the Hadsel mill at the Beebe plant near Georgetown, Calif. The mill is a duplex unit 24 feet in diameter equipped with 3-foot buckets dumping on manganese-steel plates. Maximum fall of ore is 16 feet. Maximum ore feed is 12 inches in diameter. Mill handles 209 tons of run-of-mine ore a day grinding to 71.03 percent through 200-mesh, or 308 tons a day 65.2 percent through 200-mesh.

137. ENGEL, A. L. Advantages of washing flotation feed. Min. and Met., vol. 12, 1931, pp. 234-235.

The presence of soluble salts, acids, colloids, etc. in flotation pulps has been recognized as contributing to poor results. Conditioning reagents, such as lime, soda ash, etc., are added to overcome these effects. The possibility of washing the ground flotation pulp with water (and thickening) before flotation is here discussed. This resulted in improved results in the laboratory.

138. _____ Some aspects of ore dressing. Min. and Met., vol. 12, 1931, pp. 447-449.

General observations on daily operations at milling plants from the operator's standpoint. Many practical suggestions as to plant layout and operation are made, and many errors to be eliminated are noted.

139. ERCHIKOVSKII, G. O. Enriching Khibinsk apatite by flotation. Udobrenie Urozhai (Fertilizers and Crops), vol. 2, 1930, pp. 109-114.
With crude birch tar (1.3 kilograms, together with 0.3 kilogram potassium hydroxide per ton of apatite) more than 85 percent of apatite was extracted. Good results were obtained by using 0.6 kilogram birch tar, 0.15 kilogram potassium hydroxide, and 0.34 kilogram waterglass per ton. Experiments with oleic acid (0.7 to 0.13 kilogram per ton of apatite) gave even better results than birch tar. Guaiacol, a birch-tar compound, also gave very good results.

140. FAHRENWALD, A. W. Solubility, peptization, wetting, and flotation. Trans. Electrochem. Soc. (Prepr. 60), 1931, 6 pp.
The chemistry involved in flotation is surface chemistry. The mechanism of mineral sinking is understood less well than the mechanism of mineral floating. Flotation is effected when the energy transition from phase to phase is abrupt. Little is known accurately of the heat of wetting minerals by water. Peptization is the result of a strong chemical bonding force between the solid surface molecules or atoms and water molecules or ions in the water phase. Sphalerite is weakly peptized in water but strongly peptized in very dilute solutions of sodium cyanide. Peptization falls off with increased concentration of sodium cyanide. There appears to be a relation between peptization, wettability, polarity, and flotation concentration.

141. FAHRENWALD, A. W., and ABELE, L. T. Effect of grinding time and pulp density on flotation. Eng. and Min. Jour., vol. 131, 1931, pp. 169-172.
Presents graphically a detailed study of the effect of grinding dilution and time on flotation of deslimed lead-zinc ore, other factors remaining constant. A grinding dilution of 3 parts ore to 1 part water and grinding time of 32 to 36 minutes were found most satisfactory.

142. FERET, L. R. Particle size of pulverulent materials. New Internat. Assoc. Test. Materials, September 1931, 9 pp.
The "mean width" of a particle is defined as the mean, for a sufficient number of particles, of the distances between two tangents on opposite sides of the apparent outline of the particle, parallel to an arbitrarily fixed direction. The whole range of granular and pulverulent materials is classified by limiting values of the mean widths of the various classes.

143. FICAI, CALESTINO. Process for the hydrometallurgical production of copper sulphate from Italian copper ore. Experimental study and industrial application. Ann. Scuola Ing. Padova, vol. 5, 1929, pp. 146-161.
A report of studies made to select a hydrometallurgical process suitable for the copper ore of Spezia. Especial attention was paid to roasting and lixiviation from the point of view of recovery. Semi-plant operations, as well as full-scale plant operation, are described.

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144. FIELDNER, A. C. Some experimental data on the influence of dry and wet cleaning on coke properties and on gas and byproduct yields of Pittsburgh and Mary Lee coals. Rept. of Investigations 3114, Bureau of Mines, 1931, 9 pp.
Details of the study of the effect of wet and dry cleaning of coals on coking properties.

145. FINK, COLIN G., and JENNESS, LESLIE G. Extraction of tantalum and columbium from their ores. Am. Inst. Min. and Met. Eng. Tech. Pub. 379, 1931, 18 pp.
Describes new leaching process using cheap reagents for extraction and separation and tantalum and columbium from ores and concentrates.

146. FOOTE, FREDERICK W. Newfoundland's active mines. Eng. and Min. Jour., vol. 131, 1931, p. 167.
Contains a brief description of the Buchans mine and 500-ton mill on Bell Island, Newfoundland. The ore consists of sulphides of lead and zinc in a barite gangue. Ore reserves and assays are given, as well as a summary of milling operations for 1929.

147. FORRESTER, D. L., and CRAMER, W. B. Milling methods and costs at the concentrator of the Old Dominion Co., Globe, Ariz. Inf. Circ. 6467, Bureau of Mines, 1931, 30 pp.
The Old Dominion concentrator is a 1,600-ton straight flotation plant at Globe, Ariz. The original mill was a 300-ton straight gravity-concentration plant placed in operation in 1905. A new plant of 800 tons capacity, employing gravity and flotation concentration, was built in 1914. This plant was converted into a straight flotation plant in 1923 and its capacity subsequently raised to about 1,600 tons a day. The ore treated consists of chalcopyrite and pyrite, with minor quantities of bornite, chalcocite, and cuprite in a gangue of quartzite, limestone, and diabase. Under present operations the ore is reduced in gyratory, disk, and roll crushers to 3/4-inch, ground to 48-mesh in ball mills (rod mill on one section) in two stages and treated in Forrester flotation machines in four stages. Middlings are reground, and primary flotation machines make finished concentrates. In 1930 the ore treated averaged 2.37 percent copper, concentrates produced 24.43 percent copper, ratio of concentration was 12.36 into 1, and sulphide copper recovery was 93.33 percent. Flotation reagents used were: Lime, ethyl xanthate, pine oil, and reconstructed oil (Scotch blast-furnace oil, pine oil, and sulphur).

148. FREUNDLICH, HERBERT. The physical chemistry of flotation. Trans. Electrochem. Soc. (Prepr. 60), 1931, 6 pp.
The phenomena involved in flotation are interpreted on the basis of capillary chemistry. Particular emphasis is laid on the differences in wettability of mineral and of gangue. Besides such factors as wettability, adsorption, and formation of monomolecular surface layers the time factor is very important. The stability of a good commercial froth depends largely upon the more or less solid layers of mineral particles on the surfaces of the bubbles of the froth.

149. FUJII, MITSUZO. Measurement of fine particles of cement by means of air analyzer. Jour. Japanese Ceram. Assoc., vol. 39, 1931, pp. 788-801.
Mathematical study of the range of application of Stokes' law and general consideration of methods of determining particle sizes.

150. FULOP, JEAN. Treatment of low-grade gold ores at West Rand (South Africa). Génie civil, vol. 98, 1931, pp. 153-158.
Modern equipment has raised the gold recovered by the cyanide process from 49.78 to 61.93 percent. The gold in the tailings is lowered from 0.316 to 0.256 pennyweight per ton.

151. GALLOWAY, J. W. A new type of screen. Canadian Min. Jour., vol. 53, 1932, pp. 360-361.
Describes a new type of screen, which, as a result of a new method of fabrication, presents an almost perfectly smooth surface to the flow of rock, resulting in more uniform wear giving a longer life.

152. GANDRUD, B. W., COE, G. D., and THOMAS, M. F. Washability data on certain coal beds of Alabama, with special reference to sulphur elimination. Rept. of Investigations 3157, Bureau of Mines, 1932, 28 pp.
The washability of the Jefferson, Corona, Pratt, and Carter coal beds in the Warrior field of Alabama with respect to ash and sulphur reduction is tabulated in detail. Elimination of ash is satisfactory in all. Even after crushing to 3/16-inch and tabling, sulphur in the coal from the Jefferson and Corona beds, and probably the Carter bed, is too high to meet the requirements of the byproduct-coke industry. From the Pratt bed a washed coal of 4.5 percent ash and 1.0 percent sulphur can be made easily by tabling.

153. GAUDIN, A. M. Flotation reagents. Eng. and Min. Jour., vol. 132, 1931, pp. 70-71.
Flotation reagents are divided into groups, such as frothers, collectors (or promoters), activators, depressants (or inhibitors), and alkalinity-controlling agents. Frothers (pine oil, cresylic acid, and eucalyptus oil) are substantially insoluble, have polar nonpolar molecules, and form films around air bubbles, stabilizing the froth. Collectors, or promoters, are of two classes, collecting oils (such as coal-tar creosote, wood tar, etc.) and chemical collectors (such as xanthate, dithiophosphate, etc.). These reagents form films around mineral particles, causing attachment to the air bubbles. Activators react with mineral surfaces, rendering the mineral more amenable to flotation; copper sulphate is the most commonly used activator. Depressants are of two groups, organic colloids (glue, gelatin, and casein) and inorganic salts (lime, zinc sulphate, sodium sulphide, sodium sulphate, cyanides, dichromates, and silicates). These salts can hydrate mineral surfaces (OH reactions), form ionic complexes (CN reactions), or precipitate insoluble coatings (O reactions). Many depressants are specific for certain minerals. Alkalinity-controlling reagents, such as lime, soda ash, etc., are very important in flotation circuits, and pH control is necessary.

154. GAUDIN, A. M. Some physicochemical problems of flotation. Trans. Electrochem. Soc. (Prepr. 60), 1931, 9 pp.
An outline is presented of the principal physicochemical problems encountered in flotation, particularly those dealing with the production of gas bubbles, character of the phases involved, modification of the surface of the solids of mechanics of gas-solid attachment, electrostatic effects, and economic elimination of solid-solid interfaces.

155. GAUDIN, A. M., GROH, J. O., and HENDERSON, H. B. Sizing by elutriation of fine ore-dressing products. Ind. Eng. Chem., vol. 22, 1930, pp. 1363-1366.
The functioning of the type of elutriator designed by Gross, Zimmerley, and Probert has been studied. Tables are given showing the rates of flow required in elutriators of a given cross section to separate a given mesh size of particle. Temperature corrections are also given for water and acetone. Nonpolar liquids like kerosene cause flocculation of most particles. Heterpolar liquids like alcohols cause flocculation of silicates but not sulphides. Acetone gives results in about one fourth the time required for water. Microscopic measurements show good agreement between actual and calculated particle size.

156. _____ Effect of particle size on flotation. Am. Inst. Min. and Met. Eng. Tech. Pub. 414, 1931, pp. 3-23.
Studies effect of particle size on flotability. In one method final products of several mills were sampled and tested, and in the other mixtures of pure minerals were ground to different degrees of fineness and floated. In both cases the products were sized and analyzed. Conclusions: The results obtained by the two methods. The usual notion about the relative ease of flotation of very fine particles does not agree with the facts. There is a time sequence of flotation with regard to size, the medium coarse particles floating first. No successful method of floating extremely fine pulps has been found. Recovery is optimum in a well-defined size range. Selection is optimum in another well-defined range. The bulk of the pulp is in the range of 100- to 800-mesh. Thirteen figures and 5 tables are shown. Data from laboratory and practice agree closely.

157. GERTH, G. Flotation of nonsulphidic minerals. Metall u. Erz, vol. 27, 1930, pp. 527-529.
Nonsulphidic minerals can be floated by addition of oleic acid or sodium oleate, but to prevent gangue materials such as quartz and mica from being floated likewise sodium silicate and an acid or acid salt must be added. Precipitation of the quartz and mica is due to formation of a silicic acid gel. In an appendix to this article, W. Petersen describes the separation of oil from sand by flotation, using sodium silicate and ferric chloride to hold back the sand particles.

158. GIESER, H. S. Experimental flotation of oxidized silver ores. Am. Inst. Min. and Met. Eng. Tech. Pub. 401, 1931, 9 pp.
Because of the other uses the number of organic sulphur compounds available for flotation is limited. Benzyl mercaptan and benzyl sulphide and disulphides show interesting results. Thioureas leave high tailings. When the diethylphenyl- and di-o-tolyl-thioureas are used the froth is white and barren. Mercaptobenzothiazole is of some value with a number of oxidized precious-metal ores. Thioacetanilide gives fair results, while thiobenzanilide is better. Solubility of reagents is an important factor in flotation.

159. GILBERT, WILLIAM. Grinding-plant research. Rock Products, vol. 34, no. 24, 1931, pp. 39-42.
General principles and fundamentals of tube-mill grinding are given.

160. GLATZEL, _. Dressing of iron pyrites and heavy spar at Meggen. Metall u. Erz, vol. 27, 1930, pp. 642-654, and vol. 28, 1931, pp. 7-14.
Discusses thoroughly dressing of these ores, including also the geology of the deposits, chronological development of their treatment, equipment used, power requirements, efficiency, water consumption, waste-water purification, and cost.

161. GODARD, J. S. Amalgamation tests on gold ore from the Granada Rouyn Mines (Ltd.), Rouyn district, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 143-154.
A siliceous ore containing 7.89 ounces gold per ton yielded 90 percent of its value by amalgamation at minus 35-mesh and 95 percent at minus 48-mesh. Tabling or flotation makes good recoveries from the tailing.

162. _____ Experimental tests on a sample of tailings from the mill of the Argonaut Gold Mines (Ltd.), Larder Lake, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 110-112.
A tailing assaying 0.115 ounce gold per ton, 0.19 percent copper, and 9.93 percent iron, the remainder mainly silica, gave a higher recovery by cyanidation (85 percent) than flotation.

163. _____ Experimental tests on cobalt ore from the Yorkshire Cobalt Mines (Ltd.), Cobalt, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 160-163.
Cobaltite occurs in fine crystals in a diabase gangue assaying (2 lots) 2.19 and 2.70 percent cobalt; 0.04 and 0.03 percent nickel; 2.85 and 3.55 percent arsenic; and 1.18 and 0.46 ounces silver per ton. Flotation gave cobalt recoveries of 78 and 83 percent. The ore is slow to respond to flotation. Tabling the tailings gives a low-grade product.

164. GODARD, J. S. Experimental tests on copper-bearing gold ore from the Central Manitoba Mines (Ltd.), Wadhope, Manitoba. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 137-139.

Free gold (0.30 to 0.64 ounce) associated with chalcopyrite and pyrite occurs in a siliceous gangue. Amalgamation, followed by the flotation of the tailing gave gold recoveries of 90 to 95 percent. Flotation, with tabling of the tailings recovered 94 percent of the copper. Flotation to remove chalcopyrite, followed by cyanidation of the tailing, gave 97-percent recovery (60 percent by cyanidation).

165. _____ Experimental tests on gold ore from the Arrowhead Consolidated Mines (Ltd.), Joannes Township, Rouyn Area, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 24-29.

A gold ore containing 1.21 ounces per ton was floated and the tailing tabled, amalgamated and the tailing floated, amalgamated at various sizes, and cyanided at various sizes. At minus 100-mesh 98 to 99 percent can be recovered by cyanidation, with consumption of 0.4 to 0.5 pound of potassium cyanide and 3.5 to 4.4 pounds of lime per ton.

166. _____ Experimental tests on gold ore from the Beardmore Gold Mines (Ltd.), Port Arthur Area, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 11-17.

Amalgamation of an ore assaying 0.93 ounce gold and 0.13 ounce silver per ton and 0.03 percent lead, 0.02 percent zinc, 0.02 percent copper, 4.12 percent iron, 1.03 percent arsenic, and 79.51 percent insoluble gave recoveries of 57 percent of the gold at 8 percent plus 48-mesh and 79 percent at 5 percent plus 100-mesh. Flotation with potassium xanthate (0.15 pound per ton) and pine oil (0.08 pound per ton) after ball-milling with coal tar creosote (0.20 pound per ton) gave 83.6 percent of the gold in concentrates assaying 9.58 ounces gold per ton; tabling the tailing gave 13.7 percent more in a product containing 2.46 ounces gold, a total of 97.3 percent in a 6.79-ounce product. Cyanidation of minus 100-mesh ore gave recoveries of 97 to 98.5 percent, with consumption of 0.53 to 0.61 pound of potassium cyanide and 5.3 to 6.9 pounds of lime per ton.

167. _____ Experimental tests on gold ore from the Brookfield mine of the United Goldfields of Nova Scotia (Ltd.), Liverpool, Nova Scotia. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 135-145.

A low-grade gold ore assaying 0.21 ounce per ton, chiefly in arsenopyrite and pyrite, in which 40 percent of the gold is free, was treated by various methods of concentration. Maximum recovery by amalgamation was 52 percent at minus 100-mesh. Cyanidation recovered 74 to 86 percent of the gold, with consumption of 0.75 pound of potassium cyanide per ton and 3.5 pounds of lime per ton. Flotation, followed by tabling of the tailings, gave 96- to 97-percent recovery. Without roasting, 66 percent of the gold in the concentrates was recovered by amalgamation and 87 percent by cyanidation. After roasting, amalgamation recovered only 40 percent and cyanidation 95 percent, an over-all recovery of 91 percent.

168. GODARD, J. S. Experimental tests on gold ore from the Central Patricia Mines (Ltd.), Savant Lake Area, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 54-64.
Details of flotation, tabling, and cyanide tests on an arsenopyrite-pyrite-pyrrhotite-silica ore containing 0.60 to 0.82 ounce gold per ton are given. Cyanidation gave recoveries of 95 to 98 percent but with consumption of 2 to 6 pounds of potassium cyanide and 13 to 27 pounds of lime per ton of ore. Flotation gave 80- to 90-percent recovery, increased by tabling of the tailings to 93 to 97 percent.

169. _____ Experimental tests on gold ore from the O'Brien mine, Cadillac Township, Abitibi County, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 69-80.
A gold ore assaying 2.77 ounces per ton gave recoveries of 76 percent by amalgamation alone with grinding to minus 65-mesh, 96 percent by amalgamation followed by cyanidation of the tailings, 95 percent by amalgamation and flotation, and 97 percent by amalgamation, flotation, and tabling. Details of 15 tests are given.

170. _____ Experimental tests on gold ore from the San Antonio Gold Mines (Ltd.), Lissett, Manitoba. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 116-122.
A gold ore assaying 1.51 ounces per ton was treated by various methods. Cyanidation of various sizes from minus 48- to minus 150-mesh gave 92.5 percent extraction, with consumption of 0.36 to 0.47 pound of potassium cyanide and 2.36 to 3.80 pounds of lime per ton. Flotation with sodium carbonate 2.0 pounds, coal tar 0.15 pound, potassium xanthate 0.25 pound, and pine oil 0.04 pound per ton, followed by tabling of the tailings, gave recoveries of 96.5 to 98 percent. Two-stage flotation alone gave 95.8- to 96.2-percent recovery. Amalgamation recovered 74.5 percent of the gold; flotation of the amalgamation tailings (0.32 ounce gold per ton) gave a total recovery of 96.8 percent.

171. _____ Experimental tests on gold ore from the Siscoe Gold Mines (Ltd.), Amos, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 158-159.
A free-milling gold ore assaying 0.53 ounce per ton and 0.44 percent sulphur gave recoveries of 90 to 93 percent by amalgamation, with 6 to 7 percent more by flotation of the tailings.

172. GODARD, J. S. Experimental tests on ores from the Giant mine, Spillamacheen. Golden Mining Division, B. C. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 155-158.

A barite ore analyzing 0.05 percent lead, 0.10 percent iron, 0.05 percent zinc, 93.26 percent barium sulphate, 2.72 percent silica, and 0.14 ounce silver per ton by flotation gave recoveries of 90 to 94 percent of the barite in a product analyzing 95 to 98 percent barium sulphate. An ore containing galena in a gangue of barite and silica and analyzing 17.55 percent lead, 0.04 percent copper, 0.41 percent iron, 0.11 percent zinc, 43.46 percent barium sulphate, 32.74 percent silica, and 4.66 ounces silver per ton by flotation gave good recoveries of lead (90 to 95 percent) and silver (86 to 94 percent) but only 58 percent of the barite in a 93-percent product. A chalcopyrite ore analyzing 0.11 percent lead, 4.30 percent copper, 8.36 percent iron, 0.13 percent zinc, 2.25 percent barium sulphate, 70.54 percent silica, and 1.14 ounces silver per ton yielded a flotation concentrate assaying 25 percent copper and 6 ounces silver per ton, with recoveries of 93 percent of the copper and 90 percent of the silver.

173. _____ Flotation tests on a sample of cyanide tailings from the Kirkland Lake Gold Mines (Ltd.), Kirkland Lake, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 146-150.

Tailings containing 0.062 ounce gold per ton (associated with pyrite), 23 percent moisture, 0.009 percent water-soluble lime, and a trace of cyanide were floated with a gold recovery of 62.5 percent in concentrates assaying 1.46 ounces gold per ton, largely from the minus 200-mesh material.

174. _____ Flotation tests on lead-silver ore from the Mallard Lake Gold Mines (Ltd.), Kirkland Lake, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 3-4.

A low-grade lead-silver ore with chalcopyrite and sphalerite in calcite, assaying 0.02 ounce gold and 6.40 ounces silver per ton, 5.70 percent lead, 0.47 percent copper, and 0.49 percent zinc gave recoveries by flotation of 97 percent of the lead and 94 percent of the silver in concentrates assaying 65 percent lead, 4.81 percent copper, 2.13 percent zinc, 68 ounces silver, and 0.15 ounce gold per ton with soda ash, cyanide, thiocarbanilide and pine oil.

175. _____ Tests on two types of ore from the Sylvanite Gold Mines (Ltd.), Kirkland Lake, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 66-69.

The typical, reddish porphyry of the district (I) containing 0.47 ounce gold per ton and a more schisted, more siliceous sample (II) containing 0.39 ounce gold were tested. I was more refractory than II. A higher extraction and a lower tailing were obtained from II than from I by cyanidation. Sample II showed a slightly faster settling rate but consumed 28 times as much cyanide.

176. GODARD, J. S., and HALFORD, D. S. Concentration of copper ore from Patterson Copper Mines (Ltd.), Boston Creek, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 5-6.
The ore, containing chalcopyrite and pyrite in siliceous gangue, analyzed 8.51 percent copper, 12.10 percent iron, 4.75 percent lime, 45.5 percent insoluble, and 0.22 ounce silver per ton. The ore concentrated readily by flotation, yielding 28.1 percent copper concentrates, with a recovery of 98.3 percent of the copper with 1.0 pound soda ash, 0.1 pound thiocarbanilide, and 0.1 pound cyanide per ton.

177. GOETZ, CARL, and HOROVIC, ALEXANDER. The scientific fundamentals of colloidal ore dressing. Internat. Bergwirt. Bergtech., vol. 24, 1931, pp. 37-42.
Reviews elementary theoretical principles relating to suspensoids. The peptization of a complex Rammelsberg ore analyzing 19.67 percent silica, 24.63 percent sulphur, 16.31 percent lime, 8.3 percent copper, 5.17 percent lead, 17.28 percent iron, and 5.28 percent zinc and pulverized to 10,000-mesh was studied by shaking with citric acid solutions for definite periods, settling, and determining the quantity in suspension. The maximum quantity of ore remaining in colloidal suspension depends on the citric acid concentration and on the ratio of ore to peptizing solution. The results agree with Ostwald's precipitation rule.

178. GOLDMANN, __. Mechanical conveying equipment for solid materials. Chem. Fabrik, 1931, pp. 422-424, 430-431, 437-440, 446-448, and 454-456.
Describes and illustrates various types of conveyors, elevators, hoists, cranes, trucks, cables, dump cars, electromagnets, etc., of German manufacture.

179. GONELL, H. W. A new method of grinding. Arch. Warmewirt, vol. 11, 1930, pp. 104-106.
Describes new mill for extremely fine grinding. A short, vertical cylinder ending in a bowl makes about 1,200 r.p.m. Numerous small steel balls crush the charge. Air classification is used.

180. GOODWIN, W. M. Cyaniding 40-mesh pulp at Red Lake, Ontario. Eng. and Min. Jour., vol. 132, 1931, pp. 109-110.
Describes 500-ton-capacity cyanide mill at Red Lake, Ontario, treating a siliceous ore containing about $6.00 a ton in free gold and carrying minor quantities of sulphide minerals. A flow sheet is given. Total mining and milling costs are about $3.50 a ton.

181. _____ Canada's oldest producing copper mine: The Eustis mine of the Consolidated Copper & Sulphur Co. Canadian Min. Jour., vol. 52, 1931, pp. 571-576.
Discusses earlier operations at the Eustis mine and mill and gives a flow sheet of the present flotation plant and metallurgical data. The pyrite-chalcopyrite ore is treated by flotation, using soda ash and sodium cyanide in the ball mill. Copper and iron concentrates are made.

182. GOOSKOV, W. Float and sink tests. The use of clayey suspensions as a medium. Coll. Guard., vol. 142, no. 3670, 1931, pp. 1531-1532.
The article discusses the disadvantages of using solutions of zinc chloride and calcium chloride of various densities as the medium for float-and-sink testing, describes results obtained by using kaolin suspensions, and concludes that more satisfactory results are obtained by the latter method.

183. GÖTTE, A. Concentration of tin ores in Cornwall. Internat. Bergwirt. Bergtech., vol. 24, 1931, pp. 219-225.
Gives flow sheets and practical operating details for four Cornwall tin mills.

184. ______ Present status of the dressing of bituminous copper ores. Metall u. Erz, vol. 28, 1931, pp. 73-75.
Discusses use of the flotation process in treating bituminous copper ores from Niedermansberg and Mansfield, Germany.

185. GOUDGE, M.F., and GODARD, J. S. Experimental tests on the possible use of an occurrence of dolomitic limestone near the Howey gold mine in the cyanide treatment of Howey gold ore. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 180-184.
A dolomitic limestone occurring near the Howey mine was burned to a lime which gave a higher available alkalinity than commercial lime and could be used in the cyanide circuit at the Howey mill.

186. GOW, A. M., and GUGGENHEIM, M. Dead-load ball-mill power consumption. Eng. and Min. Jour., vol. 133, 1932, p. 632.
Using a 6-foot by 4-foot ball mill with wooden bulkheads and a direct-current motor, the total power input at 80 percent of critical speed was 87 hp. and the dead-load 11 hp. With a 90-percent motor efficiency the total power available for crushing was 67.3 hp. and the total power loss 23 percent.

187. GRANT, JULIUS. Physical chemistry in relation to separation of ores. Crushing and Grinding, vol. 1, 1932, pp. 134-135.
Principal factors in froth flotation are: (1) Surface tension of the medium, (2) size of particle, (3) agitation and the tendencies of the system toward coagulation or adsorption, (4) foreign substances, and (5) alkalinity. Flotation phenomena can be explained electrically. These points are discussed.

188. GRANT, L. A. Milling methods and costs at the Spring Hill concentrator of the Montana Mines Corporation, Helena, Mont. Inf. Circ. 6411, Bureau of Mines, 1931, 8 pp.

The Spring Hill mill is a straight flotation plant of about 200 tons capacity built in 1929, treating gold ore containing pyrrhotite, pyrite, and arsenopyrite in a dioritic gangue. Most of the gold is associated with the pyrite and arsenopyrite; the pyrrhotite is barren of gold values. The plant originally was built as a cyanide mill but was converted to flotation due to contamination of cyanide solutions by arsenic. The ore is reduced to minus 1-inch in two jaw crushers and a cone crusher in series. Grinding is in conical ball mills, in closed circuit with rake classifiers. Flotation is done in Minerals Separation machines, using pine oil, ethyl xanthate, Aerofloat, and copper sulphate. The ore treated assayed $6.46 in gold, concentrates 3.24 ounces gold, and tailings $1.22 in gold, representing a gold recovery of 82 percent. The report describes the equipment used, gives flow sheets, metallurgical and cost data, etc.

189. GRANT, L. A., and PIERCE, A. L. Mill and surface improvements at the Spring Hill mine. Min. Cong. Jour., vol. 17, 1931, pp. 383-388.

Describes in detail the crushing, grinding, and flotation equipment used at the Spring Hill mill near Helena, Mont., giving flow sheets, plant layouts, and metallurgical data.

190. GROSS, JOHN, and ZIMMERLEY, S. R. A device for determining work input to a laboratory ball mill. Rept. of Investigations 3056, Bureau of Mines, 1931, 3 pp.

Describes a spring and extension tape device which was attached to a small laboratory ball mill to measure the exact power input to the mill itself.

191. GUCHI, KICHIRO YAMA. Studies on the mineral grains in relation to ore dressing. Jour. Faculty Eng. Tokyo Imp. Univ., vol. 19, 1931, pp. 263-388.

Covers relation of particle size and shape to screen apertures and true determination of screening efficiency.

192. GWINN, J. W., and HANDY, R. S. Milling practice at the Bunker Hill and Sullivan. Min. Cong. Jour., vol. 17, 1931, pp. 609-617.

The article traces the development of ore-dressing practice from the first mill in 1886 to the present plants. The gradual replacement of equipment led to the adoption of a combined gravity and selective flotation concentration process. Mineralogical analysis of the ore is given, and metallurgical data are indicated. Flow sheets and other information are included.

193. HAAS, M. C. Flotation-cyanidation supplants all-cyanidation in Russia. Eng. and Min. Jour., vol. 133, 1932, pp. 511-515.

At a 600-ton mill treating gold ore from the Kochkar mines near Chilia, in the southern Ural Mountains, a combination of amalgamation and flotation followed by cyanidation of the concentrates has replaced an amalgamation-cyanidation process. In the previous process arsenopyrite fouled the cyanide solutions in the sand-leaching division, and primary slime greatly reduced the capacity of the slimes-leaching division. Under present conditions the ore is ground to 48-mesh in fresh water and amalgamated, recovering 40 percent of the gold. Amalgamation tailings are treated in Fahrenwald flotation machines, using soda ash, pine oil, and ethyl xanthate. The ratio of concentration is 35 into 1, and 55 percent gold recovery is made. The concentrates are reground and treated with strong cyanide solution, effecting a 98-percent extraction. Over-all recovery is about 94 percent. Simplification of the flow sheet is a major advantage, and lower costs have resulted.

194. HAMILTON, E. M. Cyanides in the metallurgy of gold and silver. Trans. Electrochem. Soc. (Prepr. 60), 1931, 7 pp.

Briefly outlines the cyanide process as applied today to gold and silver ores. The behavior of double cyanides is discussed. The cyanidation process appears to have passed its peak for gold but not for silver. Cyanidation of gold ore after flotation requires only slightly less cyanide per unit weight of gold recovered than cyanidation without flotation.

195. HAMILTON, R. M. P. A new step in cyanidation. Eng. and Min. Jour., vol. 132, 1931, pp. 809-810.

The recent development in cyanidation, known as the Geco precipitation and cyanide-recovery process, is claimed to overcome the inefficiencies of precipitation from fouled solutions. Major proportions of mill solutions may be bled off by a simple, easily operated system of cyanidation, providing maximum efficiency in recovery of values and net consumption of cyanide. The process may be applied to all the solutions before zinc-dust precipitation or to the barren solutions, or to that portion of either which may be bled from the main circuit to keep it "active" or efficient in its use of cyanide. Benefits of the process are determined by simple laboratory tests.

196. _____ Group research among gold producers. Eng. and Min. Jour., vol. 131, 1931, pp. 356-358.

A plea is made for cooperation by producers in Northern Ontario. Direct operating costs and metallurgical results of nine mills are tabulated, and a flow sheet is shown for a 500-ton combination flotation and cyanide plant. Some of the successes of the group-research plans in the Transvaal and Western Australia are mentioned.

197. HAMILTON, R. M. P. Precipitation and cyanide recovery process developed by General Engineering Co. Eng. and Min. Jour., vol. 132, 1931, pp. 53-54.

An efficient, cheap method of recovering cyanide and precious metals (if any) in barren solutions fouled by contact with cyanide-consuming ores is described. A flow sheet is shown for a precipitation and cyanide-recovery plant, and a table for precipitation and cyanide-recovery process chemical equivalents. The process is cyclical and requires hydrocyanic acid from the precipitation of gold and silver with zinc and the lime necessary to "fix" the regenerated hydrocyanic acid. Over 95 percent of the solutions going to the recovery plant may be reused as fresh water in mill circuits.

198. HANCOCK, R. T. The distribution of energy in crushing. Min. Mag. (London), June 1932, pp. 337-343.

A theoretical consideration of some of the factors involved in determining work done in crushing including a review of work done by Gross and Zimmerley.

199. HANSEN, M. S. Perfection and use of the 50-gram flotation cell. Eng. and Min. Jour., vol. 133, 1932, pp. 28-29.

Describes the development of a 50-gram cast-iron laboratory flotation cell completely covered with acid-resisting enamel. This coating effectively eliminates the objectionable features of the sheet-iron and celluloid cells of Gates and Jacobson. Details of operation of the machine are given.

200. HARDY, T. W., and BLEAKNEY, H. H. Laboratory concentration of Wabana iron ore. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 195-198.

Even after grinding to minus 200-mesh, 41.6 percent of the insoluble and 42.8 percent of the phosphorus are still locked with the iron and retained in concentrates, making them too low-grade for sponge iron.

201. _____ Laboratory concentration of Bell iron ore. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 198-200.

Magnetic concentration rejected 60 percent of the insoluble at minus 20-mesh and 77 percent at minus 200-mesh. In commercial practice the sponge iron should contain 80 to 85 percent iron.

202. HARKINS, WM. D., and GANS, DAVID M. Absorption method for the determination of the area of a powder. Jour. Am. Chem. Soc., vol. 53, 1931, pp. 2804-2806.

Describes method of measuring surface of powders by determining the amount of oleic acid adsorbed from standard solutions in the form of a monomolecular film.

203. HARRIS, C. M. Mining and metallurgical practice at Kalgoorlie, Western Australia. Chem. Eng. and Min. Rev., vol. 23, 1931, pp. 411-414.

An outline of recent improvements and planned changes at the mine and mills of Lake View & Star (Ltd.).

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204. HARSH, EARL C. Improved grinding methods in the Lehigh Valley district. Rock Products, vol. 35, no. 10, 1932, pp. 31-33.

A new-type mill, operating on the principle of a ball bearing, grinds cement rock to 90 percent passing a no. 200 sieve, with a power consumption of 9.3 kilowatt-hours per ton.

205. HASKELL, R. M. Flotation practice at the Calumet & Hecla. Min. Cong. Jour., vol. 17, 1931, pp. 528-530.

Briefly reviews history of flotation as employed at the various mills of Calumet & Hecla Consolidated Copper Co. and discusses flow sheets, cost and metallurgical data, reagents, etc., now prevailing.

206. HATCH, W. G. Recent changes in milling practice at Britannia. Canadian Min. and Met. Bull. 244, Transaction Sec., 1932, pp. 323-334.

The flow sheet of the Britannia mill is described. Forrester flotation cells have been substituted for Minerals Separation machines. The advantages of the change include savings of approximately 0.10 pound pine oil, 0.6 kilowatt-hour, and 0.30 cent maintenance charge per ton of ore, 1 man per shift, less copper loss in tailings, increase of 1.5 percent in the grade of concentrates, and increase in machine capacity per unit of floor space. Present and past operations are compared in detail.

207. HAULTAIN, H. E. T., and JOHNSTON, W. E. Is the gold in tellurides soluble in cyanide? Canadian Min. and Met. Bull. 243, 1932, pp. 349-352.

Cyanide and lime solution dissolved in 48 hours over 99 percent of the gold in the particular telluride treated. Conflicting operating reports are given.

208. HEAD, R. E., CRAWFORD, A. L., and THACKWELL, F. E. An application of microscopy to an unusual manganese ore-dressing problem. Proc. Utah Acad. Sci., vol. 8, 1931, pp. 131-134.

After other means had failed it was conclusively proved by means of the microscope, aided by chemical analysis, that the manganese was combined chemically with siderite. This explained previous failures to concentrate the manganese and suggested that (1) purified manganiferous siderite concentrate be made and shipped direct to steel plants, where it could be utilized as a self-fluxing ore in making up blast-furnace charges; (2) the manganiferous siderite concentrate might be roasted to produce spiegeleisen.

209. HERBST, FRITZ. The modern Ramsbecker mine. Metall u. Erz, vol. 29, 1932, pp. 194-196.

Describes ore dressing at the Ramsbecker mine.

210. HERZ, NATHANIEL. Laboratory shaking machine. Eng. and Min. Jour., vol. 133, 1932, p. 22.

Describes 32-compartment shaking machine for amalgamation and cyanidation testing.

211. HILL, H. R. The East Geduld Mine. Min. Mag. (London), February 1932, pp. 73-85.

Discusses in detail cyanide plant at East Geduld property in Far East Rand. Includes flow sheets and metallurgical data.

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212. HINTZE, F. F., and LANGE, L. H. Separation of cyanite and mica from quartz, feldspar, and other gangue minerals of a mica schist. Rept. of Investigations 3085, Bureau of Mines, 1931, 6 pp.

Cyanite and mica can be recovered by crushing, screening, and tabling. For the highest purity of concentrates additional screening and water classification might be employed. Concentration should be done with as coarse a feed as possible. Success in tabling depends upon close sizing of the ore. Very little cyanite was found in the minus 100-mesh material.

213. HODGES, FRED. Milling methods at the Hurley plant of the Nevada Consolidated Copper Co., Hurley, N. Mex. Inf. Circ. 6394, Bureau of Mines, February 1931, 16 pp.

The original Hurley milling unit was placed in operation in 1911 and was a straight gravity-concentration unit using tables and vanners. Flotation of slimes was begun in 1914, and the vanners were discarded in 1923. All gravity-concentration equipment was replaced with flotation machines in 1927. Lime was adopted in 1927. The ore treated consists of chalcocite, with minor quantities of oxidized copper minerals, associated with pyrite in a gangue containing quartz, sericite, and halloysite. Under present concentrating methods the ore from the open-pit mine is reduced to minus 4 inches in jaw and gyratory crushers at the Santa Rita mine and transported in 55-ton cars to Hurley. Intermediate crushing at the mill is done in cone and roll crushers and grinding in two stages, using rod mills and ball mills. Flotation is carried on in air-lift-type machines; lime, pine oil, and raconite are employed. The mill treats ore averaging 1.32 percent copper and recovers 83.98 percent of the copper in concentrates averaging 27.54 percent copper. The report gives flow sheets, metallurgical data, etc.

214. HOMER, D. D. Rubber pipe lining minimizes pulp abrasion. Eng. and Min. Jour., vol. 132, 1931, pp. 367-368.

A metal pipe lasted for the passage of 1,000 tons of tailing, and extra-heavy metal pipe lasted for 2,000 tons. A rubber-lined pipe showed negligible wear after the passage of 250,000 tons.

215. HOWBERT, VAN DYNE, and GRAY, FRED E. Milling methods and costs at Presidio mine of the American Metal Co. of Texas. Am. Inst. Min. and Met. Eng. Tech. Pub. 368, 1930, 20 pp.

Oxidized and siliceous ore is treated by gravity concentration and cyaniding. An average analysis of the ore is given as 19.7 ounces silver, 0.02 ounce gold, 2.5 percent lead, and 4.87 percent zinc. The ore is crushed and the minus 10-mesh material tabled to produce silver-lead concentrates analyzing 296 ounces silver, 0.52 ounce gold, and 43.4 percent lead. The tailings are combined with the plus 10-mesh material to be reduced in a tube mill in circuit with a classifier. The classifier discharge, which is 97 percent minus 100-mesh, is cyanided. The combined recoveries amount to 90.3 percent of the silver, 97 percent of the gold, and 34.3 percent of the lead. Cost data are given for the various steps in the process.

216. HOWE, R. E. Recent improvements at Cananea. Arizona Min. Jour., vol. 14, 1931, pp. 8-9.

Describes new crushing plant and changes in concentrator at the Cananea mill.

217. HUBER-PANU, ION. Consideration of the fundamental curves of ore-dressing processes. Metall u. Erz, vol. 28, 1931, pp. 549-551.
A mathematical formula for the course of ore-dressing processes is developed, from which expressions can be derived for calculation of concentrate content, metal yield, and separation coefficient.

218. _____ Influence of temperature upon flotation. Metall u. Erz, vol. 28, 1931, pp. 545-549.
At 40° flotation time is only about half that at 6°. With rising temperature the total yield increases until a maximum is reached and then begins to fall. Each ore and method used has its optimum temperature, generally 23 to 40°. Decrease in yield at higher temperatures is due to increased oxidation of the mineral surface and increased solubility of the compounds formed at this surface.

219. HUBLER, W. G. Flotation at the Amulet mine, Quebec. Min. Mag. (London), May 1931, pp. 305-309. (Reprint of article in Canadian Min. and Met. Bull. 236, 1931, pp. 295-306.
Describes in detail equipment and operation of 350-ton Amulet flotation mill, treating complex copper-zinc-iron ore. Mineralogical analysis of the ore is stated and the action of the various flotation reagents fully discussed. Flow sheets and metallurgical data are included.

220. _____ Pyrite flotation at Aldermac, Quebec. Canadian Min. and Met. Bull. 239, 1932, pp. 82-91.
Discusses flotation reagents used in treating copper-iron ore from Aldermac mine. On mine ore reagents used in copper circuit were soda ash, cyanide, thiocarbanilide, pine oil, and ethyl xanthate. In the pyrite circuit copper sulphate, ethyl xanthate, and water-gas tar were used.

221. HUNT, H. D. Milling methods and costs at the concentrator of the Miami Copper Co. Miami, Ariz. Inf. Circ. 6573, Bureau of Mines, 1932, 25 pp.
The ore treated at this large concentrator is of the so-called disseminated porphyry type, consisting of chalcocite and pyrite (with minor quantities of oxidized copper minerals) in a gangue of silica, feldspar, muscovite, biotite, and small amounts of magnetite. The first milling plant was a straight gravity-concentration unit, placed in operation in 1911. Flotation was tried experimentally in 1913, and by 1915 all sections of the mill had pneumatic machines handling primary slime. Early in 1919 all gravity-concentration equipment had been discarded, and the plant was operating as a straight flotation plant. Changes and improvements in the crushing, grinding, and flotation sections were made during this period. Alkaline circuits using pine oil and xanthates were introduced in 1924. The entire plant was rebuilt to its present form in 1928 and 1929. The present method of milling includes crushing in cone crushers and two sets of rolls in series, three-stage grinding in ball mills and classifiers, rougher bulk flotation of copper and iron in 100-foot Miami pneumatic cells, regrinding, and preferential flotation of rougher concentrates to make copper concentrates assaying more than 34 percent copper from ore assaying 0.604 percent copper (as sulphide), with a ratio of concentration of 60.4 into 1 and a recovery of 88.76 percent of the sulphide copper. The report gives many interesting details of historical development, flowsheets, equipment, metallurgical and cost data, etc.

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222. HUTTL, J. B. Tungsten mining and milling. Eng. and Min. Jour., vol. 131, 1931, pp. 504-506.

A description of the operations of the Tungsten Alloy Corporation in the Las Guijas Range in southern Arizona. Gravity concentration is employed, treating ore assaying 1.5 to 2.0 percent tungstic acid. A recovery of 80 percent is made in concentrates assaying 40 percent tungstic acid.

223. _____ Gravity milling and flotation of low-grade gold ore. Eng. and Min. Jour., vol. 133, 1932, pp. 630-631.

Describes mining and milling operations at property of Beebe Gold Mining Co. at Georgetown, Calif. Discusses Hadsel mill **briefly.** Flotation is used to treat the classifier overflow, the concentrates joining the sands and advancing to the cyanide leaching vats. Flotation reagents used are Aerofloat and amyl xanthate.

224. ISERN, ELMER. Central milling in the tri-State district. Eng. and Min. Jour., vol. 131, 1931, pp. 49-54.

A detailed description of the design, construction, and operation of a centralized mill in the tri-State region treating ore from several mines. The Bird Dog mill of the Commerce Mining & Royalty Co. has a capacity of about 115 tons of zinc and lead-zinc ore an hour and uses a combination of jigs, tables, and flotation machines. The receiving, unloading, crushing, and sampling of the ore are described and the details of the milling equipment are given.

225. _____ Compressed-air sampling. Eng. and Min. Jour., vol. 131, 1931, pp. 351-354.

Describes automatic, compressed-air actuated, sampling equipment used at the Bird Dog mill in handling ore from seven mines.

226. IVERSON, H. G. Separation of feldspar from quartz. Eng. and Min. Jour., vol. 133, 1932, pp. 227-229.

Outlines laboratory investigation of treatment of a granite containing biotite, orthoclase, and quartz. The mica was readily floatable; but the reagents used were not selective enough and much of the feldspar also floated. The mica can be removed by table concentration and the quartz-feldspar product treated by flotation to yield a commercial feldspar product. Reagents used were terpineol, sodium oleate, and lead nitrate.

227. JACKSON, CHARLES F., and KNAEBEL, JOHN B. Gold mining and milling in the United States and Canada. Bull. 363, Bureau of Mines, 1932, 151 pp.

Discusses geology, exploration, development, mining, milling, and costs of producing gold in detail. Considers factors affecting the choice of milling methods, including character of ore, tonnage, capital available, water supply, commercial affiliations, royalties, location, and general economic conditions. Describes milling methods and offers numerous examples of current practice, with costs.

228. JOHNSTON, A. McA. Effect of adding common salt to cyanide solutions. Jour. Chem. and Met. Soc. South Africa, vol. 32, 1932, pp. 154-155.
The addition of sodium chloride to the cyanide leach solution increased the gold extraction under standardized conditions. Salt also aided in the precipitation of gold from solution by zinc shavings but had no appreciable effect when added to the melting flux.

229. JONES, G. CARLETON. The Sub Nigel gold mine. Min. Mag. (London), March 1932, pp. 137-148.
Reviews company's operations on the Witwatersrand and describes present mining and milling methods. Mill employs a combination of corduroy strakes and sand and slime cyanidation. About 48 percent of the gold is recovered from the strake concentrates by barrel amalgamation. Nearly 89 percent of the ore is treated at the slimes-leaching plant and the balance at the sand-leaching plant. Flow sheets and metallurgical data are given.

230. JULLUM, HENRY. Milling at the Argonaut. Min. and Met., vol. 13, 1932, pp. 476-477.
Describes 250-ton amalgamation and gravity-concentration mill of the Argonaut Mining Co. at Jackson, Calif. Amalgamation saves 70 percent of the gold and concentrates contain 19 percent. The tailings contain 11 percent of the gold and are treated by cyanidation at the Amador mill.

231. KAMIENSKI, B. An electrostatic explanation of the phenomenon of flotation. Nature, vol. 129, 1932, pp. 59-60.
Emulsions of dielectrics in water are claimed to wet unattackable electrodes coated with dielectric oxides or hydroxides. The wetting must, however, reduce the positive charge of the unattackable electrode. Wetting is not expected to take place on attackable electrodes coated with a dielectric hydroxide. To test these theories electrodes of platinum, lead sulphide, Acheson graphite, and zinc were immersed in 50 cm^3 of 0.002 normal potassium chloride (pH equals 7.5) and their potentials determined. On adding an emulsion of 4 grams of terpineol and 4 grams xantogenicum in 1,000 cm^3 of 0.002 normal potassium chloride of the same pH the potential of the zinc electrode changed but little. The others showed a marked fall in potential. The first portion of 0.02 cm^3 of the emulsion has a profound effect.

232. ____. So-called flotation. Przemyśl Chem., vol. 15, 1931, pp. 201-202.
The negative charge of an oil drop in water emulsion neutralizes the positive charge at the contact between water and a sulphide, such as galena, or other conducting sulphide. Quartz and sphalerite carry negative charges in contact with water and do not become wetted with oil, the drops of which in the emulsion are also negatively charged. Various organic substances only lower the surface tension of water and facilitate foaming, as in soap. If to a suspension of an electrically nonconducting sulphide is added a salt with a cation which forms less soluble electrically conducting sulphide, then the nonconducting sulphide will become coated with a film of the conducting sulphide, producing a positive charge on the particle which can be neutralized by the negative charge of an oil drop. Thus, the addition of copper sulphate makes it possible for zinc blend to be wetted by oil drops of the emulsion.

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233. KANNING, E. W., HARTMAN, R. J., and CHILDS, FLOYD. Sedimentation of clays. Jour. Phys. Chem., vol. 36, 1932, pp. 2369-2382.
The size of the particles of various Indiana clays was calculated from the velocity of fall of particles, indicated by the sedimentation in definite experimental time intervals. The results are arranged in a series showing the order of percentage of large particles in clays from different deposits.

234 KARCHMER, N. K. Flotation of apatite in Russia. Eng. and Min. Jour., vol. 133, 1932, pp. 429-432.
Describes in detail 1,300-metric ton plant in Russia, treating ore containing 65 to 70 percent apatite. Standard equipment is used to crush and grind the rock to 80-mesh, and flotation is done in MacIntosh and Fahrenwald machines using oleic acid and sodium silicate. Recovery is 95 percent into concentrates containing 37.5 percent phosphorous pentoxide, which is made into superphosphate fertilizer. The ore also contains nephelite, and plans are made to recover aluminum from this mineral in the mill tailings.

235. KEISER, A. D. Crushed-stone and ore-dressing plant: A profitable combination. Eng. and Min. Jour., vol. 132, 1931, pp. 4-7.
Describes fully 800-ton zinc flotation plant of the Universal Exploration Co. at Jefferson City, Tenn. Flow sheets and plant layouts are given, and equipment used is described. Flotation reagents used are copper sulphate, xanthate, pine oil, and Barrett no. 4. About 90 percent of the zinc is recovered in concentrates assaying 60 to 65 percent zinc. Includes operating costs.

236. _____ Suspended crusher replaces rolls. Eng. and Min. Jour., vol. 131, 1931, pp. 67-68.
Describes operation of 7-inch Newhouse suspended crusher at the Bird Dog mill of Commerce Mining & Royalty Co. Test results and comparative data are given.

237. KELLOGG, A. E. Origin of flour gold in black sands. Arizona Min. Jour., vol. 14, 1931, pp. 3-4 and 49-50.
Discusses several possible explanations for origin of flour gold and unreliability of various machines designed for its recovery.

238. KENDALL, S. WILMER. Modern ball and pebble mill technic. Jour. Oil and Colour Chem. Assoc., vol. 15, 1932, pp. 66-96.
Discusses suitability of various types of ball mills, linings, balls, and accessories. Various factors affecting grinding of paints, etc., have been collected into a formula for predicting optimum speed at which a mill should be operated. Includes a few practical notes. Discussion.

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239. KIDD, ROBERT L. Flotation in the treatment of gold ores. Min. and Met., vol. 13, 1932, pp. 405-406.

Laboratory study of the floatability of gold. Three types of ores were studied: (1) Siliceous gold ore containing no base metals, (2) pyritic gold ore, and (3) oxidized gold ore from the same vein as sample (2). Author concludes that bright, free gold is readily floatable, whether or not associated with sulphide minerals. Tarnished gold will float after cleaning by intensive grinding. Combinations of flotation and gravity concentration followed by regrinding and cleaning will grade up the concentrates without loss of recovery.

240. KING, ANDREW. Some local (South African) metallurgical considerations. Jour. Chem. Met. and Min. Soc. South Africa, vol. 32, 1931, pp. 34-44.

Discusses possibility of cyanide treatment of hand-sorted waste rock from various Rand mills.

241. KING, CECIL V., and DUBLON, EMILIE. Rate of decomposition of xanthic acid. Jour. Am. Chem. Soc., vol. 54, 1932, pp. 2177-2186.

Decomposition of ethyl- and methyl-xanthic acids in the presence of acetic acid-sodium acetate and aniline-aniline-hydrochloride buffers was followed by measuring the increase in vapor pressure above the solutions. The primary and secondary salt effects are analyzed. The salt effect is shown to complicate previous measurements. The temperature coefficient is unusually high, the average value being 6.75 for 10°.

242. KIRCHNER, A. R., GALLOWAY, J. V., and SCHODER, W. P. Milling methods and costs of the Minas de Matahambre, S. A., concentrator. Inf. Circ. 6544, Bureau of Mines, 1931, 11 pp.

Describes operations at a 1,300-ton flotation plant 6 miles inland from Santa Lucia, Province of Pinar del Rio, Cuba. As constructed in 1922, it was a 600-ton combined gravity and flotation plant using tables and pneumatic flotation machines. Tables were discarded later, and with the addition of new cells the mill became a straight flotation plant. A lime circuit was adopted in 1925, and cyanide was used first in 1929. The Matahambre ore is primary chalcopyrite and pyrite associated with quartzite and shale. Crushing is done at the mine in gyratories, cone crusher, and rolls, reducing the ore to 3/8-inch, which is sent by aerial tram to the mill. Two-stage grinding is used, the primary ball mills being in closed circuit with 3-mm trommels, and the secondary ball mills operate with rake classifiers. Flotation pulp passes to 3 primary roughers; tailings go to 4 secondary roughers and 2 scavengers; concentrates go to two-stage cleaners to the filter department; all middling products return to the head of the circuit. Flotation reagents are lime, sodium cyanide, pine oil, and xanthate. Recovery is 96 percent from 4.55-percent ore into 29.2-percent concentrates.

243. KNAEBEL, JOHN B. Milling methods at the Balmat mill of the St. Joseph Lead Co., Balmat, St. Lawrence County, N. Y. Inf. Circ. 6574, Bureau of Mines, 1932, 28 pp.

The ore treated at the Balmat mill contains sphalerite, pyrite, and galena associated with tremolite and diopside, quartz, dolomite, chlorite, magnetite, hematite, and talc. Early milling of this ore employed magnetic separation methods, but flotation was introduced in 1926. The present mill, of 500 tons daily capacity, was placed in operation in May 1930 and follows the usual three-stage flotation procedure, making separate concentrates of lead, zinc, and iron. The company experimented with a four-stage process, using a so-called "insoluble circuit" ahead of the lead circuit designed to remove talc and mica. This was accomplished in a pneumatic machine; but over-all recoveries were not improved, and the process was discarded. Under present operations the ore is reduced to minus 0.3-inch in a jaw crusher, gyratory crusher, and set of rolls, ground to flotation size in a rod mill in closed circuit with a rake classifier and treated in three stages of flotation machines with cleaning and recleaning of the concentrates. Middlings are not reground. Lead-circuit reagents are thiocarbanilide, cresylic acid, xanthate, and sodium cyanide; in the zinc circuit copper sulphate, sodium aerofloat, pine oil, lime, and xanthate are used; and xanthate, copper sulphate, sodium sulphide, and pine-tar oil are used to float pyrite. Recoveries are: Lead, 66.9 percent; zinc, 82.2 percent; and sulphide iron, 66.4 percent, on ore assaying 1.62 percent lead, 12.0 percent zinc, and 15.5 percent iron. The report includes flow sheets, metallurgical and cost data, etc.

244. KNAPP, E. A. Ratio of concentration and recovery formulae. Min. Mag. (London), July 1931, pp. 15-16.

The author presents two charts, one based on the ratio-of-concentration formula and the other on the recovery formula. With the aid of these charts the operator can read the unknown directly, thus saving calculations.

245. KOEPEL, EDWARD. Flotation of native copper at Freda, Mich. Eng. and Min. Jour., vol. 131, 1931, pp. 509-512.

The Freda mill of the Champion Copper Co. was remodeled in 1929, and was converted into a combined gravity and flotation plant using flotation to treat primary slimes and reground table tailings. Rake classifiers were installed in place of hydraulic machines at the stamp-mill discharge; and about one half the overflow is circulated back to the stamps, reducing the new water requirements and thickening the flotation feed. Flow sheets and descriptions of the new equipment are included.

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246. KRAEBER, LUDWIG. Investigation on the influence of the iron content of zinc blendes on their flotability. Mitt. Kaiser-Wilhelm-Inst. Eisenforsch., Düsseldorf, vol. 12, 1930, pp. 343-352.
Flotation tests were made on blendes in a mechanically agitated cell and in a pneumatic cell. Each sample of blende was found to give a definite pH to the water, regardless of whether the water originally was slightly alkaline or acid; the value was lower for blendes of high iron content. The maximum recovery was obtained with a pH between 6 and 7, and the optimum pH appeared lower for high iron content. Very small quantities of copper sulphide speeded the rate of recovery. The quantities of potassium cyanide and zinc sulphate necessary to hold down the iron differed widely for various ores but bore no simple relation to the iron content. Conclusion: The presence of foreign materials, as well as the pretreatment, has more significance than the iron content in affecting floatability.

247. KRAUT, MAX. Floating gold on the Mother lode. Min. and Met., vol. 13, 1932, pp. 175-176.
General discussion of the application of flotation in the treatment of California gold ores with particular reference to the function of various flotation reagents.

248. KROZIL, F. Grain-size determination of powdered adsorbents. I. Kolloid Ztschr., vol. 55, 1931, pp. 25-30.
Sedimentation velocities of water suspensions are determined conveniently with a Kohn pipette device. The finer fractions of activated charcoals ranged from 0.01 to 0.05 mm, while silica gel ranged from 0.001 to 0.05 mm.

249. KUTZNER, KARL. Separation of solid and liquid substances. Chem. Fabrik, 1931, pp. 225-233, 249-250, 265-266, and 279-281.
Discusses separation of solids from liquids by settling, filtering, and centrifuging and shows several types of apparatus used in each method.

250. LAKWEY, FRITZ. Gieche's Erben enterprise. Eng. and Min. Jour., vol. 132, 1931, pp. 151-153.
General article on Gieche's operations at the Bleischarley lead-zinc mine in Upper Silesia. Describes the various plants and shows flow sheets of the lead-zinc concentrator.

251. LAWRENCE, H. M., and ROCA, E. Flotation of low-grade phosphate ores. II. Rept. of Investigations 3105, Bureau of Mines, 1931, 9 pp.
Conditioning of thick phosphate pulp with sodium sulphide followed by flotation with oleic acid will effect a separation of phosphate and quartz at reagent costs as low as 10 cents per ton of material. This is equally effective with ground or unground sands.

252. LAWRIE, H. N. Gold and silver. Mineral Ind., vol. 40, 1931, pp. 269-307.
Discusses world production, economics, legislation, and progress in mining and metallurgy, with extensive bibliographies.

253. **LEAVER, EDMUND S.** Development of cyanidation with increasing complexity of ores. Trans. Electrochem. Soc. (Prepr. 60), 1931, 6 pp.

Cyanidation of gold was early recognized as an efficient scavenger after amalgamation. During the last decade new chemical methods have been introduced for treating more complex ores by cyanidation. Tellurides of precious metals have been leached by bromocyanide, but preliminary roasting before cyanidation offers the more certain recovery method. Silver ores containing oxidized manganese minerals usually are refractory to cyanide treatment and therefore require special preliminary handling. The presence of each base-metal mineral in ores complicates the recovery of precious metals by cyanidation. Cyanidation of copper has not been practical. In some cases flotation has aided cyanidation. Removing oxygen from the pregnant solution before precipitating gold has been an important factor in the extended use of zinc. Aluminum has been used successfully instead of zinc where cyanide-soluble arsenic and antimony formed harmful combinations with it. Flotation is a serious competitor of cyanidation. Oxidized silver ores usually give the best results by cyanidation.

254. **LEAVER, EDMUND S., and WOOLF, J. A.** Copper and zinc in cyanidation sulphide-acid precipitation. Tech. Paper 494, Bureau of Mines, 1931, 53 pp.

Discusses in detail the solubility of various copper and zinc minerals in cyanide solutions and describesthe process of using sodium sulphide-sulphuric acid in cyanide regeneration.

255. _____ Factors affecting the flotation of gold in milling ores. Trans. Electrochem. Soc. (Prepr. 60), 1931, 17 pp.

The results of this research warrant the following conclusion: lime is a depressant for gold during flotation. The maximum amount of lime permissible in the flotation circuit will vary for different ores, but in general the optimum pH range is 7 to 10. Any lime in the flotation circuit will depress gold if the lime is precipitated with carbon dioxide. Therefore, foul air, smoke, or excessive carbon dioxide should be prevented from entering with the air passing to the flotation cells. Some ore pulps are placed in a better condition for flotation by the slightly acidic condition produced by carbon dioxide in the absence of lime. Sulphidizing agents are needed to aid flotation of oxidized base-metal minerals. Sodium sulphide is not commonly used, and in general its use retards the flotation of metallic gold. In certain types of highly oxidized ores, sodium sulphide may aid by dispersing the slime, which may counterbalance the retarding effect. Adding cyanide to the flotation circuit does not increase the recovery of the free gold, but in general lowers recovery by retarding flotation of the minerals containing gold and by direct dissolution of part of the gold. Proper conditions of the pulp are important in all flotation problems, particularly for the recovery of gold. High ratios of concentration are usual with gold ores, requiring good selective action by the flotation reagents. Proper selectivity cannot be obtained if there is much interference from slime or gangue. Each ore presents an individual problem and should be studied to meet each refractory feature. Amalgamation is the best method for the recovery of the coarse part of the gold in milling ores and should precede flotation or cyanidation in the treatment of such ores.

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256. LEAVER, EDMUND S., and WOOLF, J. A. Flotation of minor gold in large-scale copper concentrators. Am. Inst. Min. Met. Eng. Tech. Pub. 410, 1931, 32 pp.

Lime is an active depressant for free gold during flotation if the alkalinity of the solution circuit is pH equals 10 or above. A reaction of pH 7.5 to 10 is best. In the flotation of base-metal ores containing gold as a byproduct careful regulation of lime, avoiding excess, will increase recovery of gold. If pyrite in the ore is crushed fine enough to free the gold, most of the pyrite may be depressed by low alkalinity of lime in the flotation circuit and a high percentage of the gold floated with the copper sulphides. Ores with a low ratio of pyrite to copper sulphide, containing under $1.00 in free gold per ton gave, by control of the flotation circuit assuring low alkalinity, 25 percent copper concentrates containing more than 90 percent of the copper and more than 80 percent of the gold. Ores high in pyrite and lean in copper sulphides and gold as a nominal byproduct offer low-grade concentrates, unless the percentage of gold recovery is sacrificed. In the flotation of oxidized base-metal ores in which a sulphidizing reagent is used, sodium sulphide gives a much higher gold recovery than calcium sulphide or calcium polysulphide; calcium salts depress gold.

257. LEAVER, E. S., WOOLF, J. A., and KARCHMER, N. K. Oxygen as an aid in the solution of silver by cyanide from various silver minerals. Rept. of Investigations 3064, Bureau of Mines, 1931, 15 pp.

Cerargyrite is completely soluble in 8 hours in cold cyanide solution. Practically all the silver in stetefeldtite is dissolved in 24 hours by cold cyanide. Argentite, polybasite, stephanite, and pyrargyrite are slowly soluble under the usual conditions of cyanidation. Their rate of solution is increased materially by supplying additional oxygen. Prousite, similar in composition, did not show any material increase in extraction by use of oxygen. By a preliminary low-temperature roast, an extreme case of oxidation, the resulting calcines yield an extraction of 90 percent of the silver by cyanidation. Tetrahedrite is the most refractory mineral tested. Direct cyanidation, even with excess oxygen, gave less than 25 percent extraction. Preliminary roasting increased this figure. In general, oxygen is an aid in treating silver minerals by cyanidation. Warm solutions also increase the solution rate. Low-temperature roasting is usually effective. Usually 72 hours or more are required for maximum solution of silver by cyanidation.

258. LIPETZ, I. M. E., and RIMSKAYA, M. M. Physical chemistry of the flotation process. III. The influence of flotation reagents on selective wetting as the physicochemical characteristic of their collecting power. Tzvetnuie Metallui, 1931, pp. 990-1003.

Discusses measurement of collecting ability of various flotation reagents by determining wetting characteristics of reagents on paraffin.

259. LIPETZ, I. M. E., and RIMSKAYA, M. M. Physicochemical discussion of the flotation process and its practical applications. IV. Determination of contents of flotation agents in aqueous media by means of surface tension measurements. Tzvetnuie Metallui, 1931, pp. 1432-1442.
Discusses use of capillary manometer method of titration for determining the concentration of flotation reagents and its application in determining the composition of mixtures of reagents.

260. ______ Surface activity and surface tension as a method of investigation of flotation reagents. Tzvetnuie Metallui, 1931, pp. 594-610.
Reviews various methods of measuring surface tension and describes Rebinder apparatus. Gives results of studies of solubilities and compositions of flotation reagents.

261. LISSMAN, MARCEL A. An analysis of mechanical methods of dust collection. Chem. and Met. Eng., vol. 37, 1930, pp. 630-634.
Mathematical analysis of the principles governing the movement of fine particles.

262. LIVINGOOD, G. B. The science of ore reduction. Canadian Min. Jour., vol. 53, 1932, part I, pp. 207-212 and 262-263; part II, pp. 293-296; part III, pp. 352-355.
General review of the fundamentals of crushing describing jaw and gyratory crushers, rolls, etc.

263. LOCKE, C. E. Mineral dressing. Min. and Met., vol. 13, 1932, pp. 25-27.
A general review of ore-dressing developments during 1931, describing new equipment introduced and discussing advances made in the application of flotation in the treatment of gold ores and nonmetallics.

264. LOTTERMOSER, A., and RUMPELT, HORST. Studies on the flotation of bauxite from deposits near Bodayk (Hungary). Kolloid-Beihefte, vol. 35, 1932, pp. 372-412.
Use of orthotoluidine, thiocarbanilide, oleic acid, and pine oil in a slightly acid circuit gave optimum results in the treatment of bauxite. Sodium silicate improved results under certain conditions.

265. LOW, BELA. Determining results graphically in one-product concentration. Eng. and Min. Jour., vol. 132, 1931, pp. 524-525.
Describes a method of developing charts from which concentration data may be determined directly without calculations.

266. LOWRY, ALFRED. All-flotation at North Broken Hill (Ltd.). Proc. Australasian Inst. Min. Met., 1930, pp. 355-377.
Describes in detail operations at the North Broken Hill lead-zinc concentrator, including reagents used, metallurgical results, etc.

267. LUBMAN, N. M. Froth formation and flotation ability of powders in solutions of surface-active substances. Tzvetnuie Metallui, 1931, pp. 854-866.
Detailed study of froth-forming characteristics and flotation ability of various reagents under different conditions.

268. **MACMILLAN, A. H., and KRASE, NORMAN W.** High-speed agitation under pressure. Ind. Eng. Chem., vol. 24, no. 9, 1932, pp. 1001-1002.
Describes high-speed agitation apparatus for use with any combination of gases, liquids, and solids up to 200° temperature and 300 atmospheres pressure.

269. **MADEL, H.** Progress in the realm of ore dressing in recent years. Metall u. Erz, vol. 28, 1931, pp. 229-240.
Reviews recent developments in sieve classification, pulverizing, gravity dressing, and flotation. Special cases in the application of flotation considered include the flotation of mixtures of galena and granite, the differential flotation of a lead-zinc ore, flotation of zinc blende, and flotation and cyanide extraction of gold ores. Includes photographs and descriptions of new apparatus.

270. **MAHON, S. A., and COUNSELMAN, T. B.** Washing Mesabi iron ores. Eng. and Min. Jour., vol. 130, 1930, pp. 519-521.
Changes made from the original flow sheet involve: (1) Replacement of the "turbos" and tables with bowl classifiers; (2) introduction of secondary crushing for further liberation of silica from the coarser sizes; and (3) tendency toward the use of abrasive screening on much of the material formerly treated in the 25-foot log washers. Shows several tables, and flow sheets for the original process, for bowl classification or secondary crushing and using the Dorr washer.

271. **MARTIN, H. S.** Milling methods and costs at the Arthur and Magna concentrators of the Utah Copper Co. Inf. Circ. 6479, Bureau of Mines, 1931, 25 pp.
Describes concentration operations at the Magna and Arthur mills, which have a combined daily capacity of more than 60,000 tons of ore. Reviews milling history, beginning with the original pilot mill at Copperton built in 1903, the original Magna mill in 1906, the Arthur mill in 1910, with the various changes and remodelings, including the adoption of flotation concentration of slimes in 1914, the regrinding and flotation of gravity tailings in 1920, and the adoption of straight flotation in 1926. Gives details of old and new crushing, grinding, and concentrating equipment, and describes mineralogical character of ores treated. Includes flow sheets, metallurgical and cost data, flotation reagents, etc.

272. McKENZIE, M. R., and LANCASTER, H. K. Milling methods at the concentrator of the Walker Mining Co., Walkermine, Calif. Inf. Circ. 6555, Bureau of Mines, 1932, 11 pp.

The ore treated at this mill contains 12.5 percent metallic minerals, consisting chiefly of magnetite, chalcopyrite, pyrite, pyrrhotite, and chalcocite. The nonopaque minerals are principally quartz, garnet, and chlorite. The ore contains 1.687 percent copper, 9.0 percent iron, 79.2 percent insoluble, and 0.05 ounce gold and 0.833 ounce silver to the ton. Milling operations were begun in 1916, with a 75-ton pilot plant using straight flotation. The present mill was placed in operation in 1923 at 750 tons capacity, which was increased to 1,700 tons a day in 1929. The mine ore is reduced to minus 1-inch by jaw crushers and rolls in series. Ball mills are in closed circuit with classifiers and grind the ore to flotation size, about 12 percent plus 48-mesh, a comparatively coarse feed. Flotation, both roughing and cleaning, is done in Callow cells. Reagents are lime, pine oil, and ethyl xanthate. Concentrates assay 24.0 percent copper, and recoveries are: Copper, 91.2 percent; gold, 62.7 percent; and silver, 76.3 percent. The report gives details on equipment, metallurgical and cost data, etc., and a flow sheet.

273. MEISSNER, H. Dressing experiments with oxysulphidic copper ores in Chuquicamata. Metall u. Erz, vol. 29, 1932, pp. 174-177.

A study of various flotation agents including alkaline ethyl and amyl xanthates, sulphides and polysulphides. The best results were obtained with sodium sulphide.

274. MEYER, H. Activity of working solutions in the cyanide process. Metall u. Erz, vol. 28, 1931, pp. 261-280.

A study of the chemical and physical processes in the cyanide extraction of ores shows that the laws of Fick, Nernst, Walden, Stokes and Kohlrausch governing diffusion processes in hydrometallurgical hetergeneous reactions are applicable. The practical consequences of this from the viewpoint of purity of solutions, rapidity of diffusion, etc., are discussed. Increase in temperature of the liquor gives an increase in reaction velocity of 1.5 to 2.7 percent per degree, the maximum reaction velocity being attained at 80°. However, since parallel secondary reactions must be reckoned with, practical considerations have indicated that the temperature for solution of gold has an upper limit of 50 to 55°. Above this, very rapid reduction by ferrocyanide and considerable loss by hydrolysis occur. The protective alkali concentration should be 0.002 mole of sodium hydroxide. With increasing concentration of cyanide the increase in extraction is approximately logarithmic. The relation between time of treatment and extraction is not linear, for extraction is a function of the cube root of time of treatment.

275. MILLER, T. H., and KIDD, R. L. Flotation reagents, 1929. Rept. of Investigations 3112, Bureau of Mines, 1931, 20 pp.
Gives tabular summary of the consumption of flotation reagents at plants in the United States during 1929. The ores treated are divided into classes and the per-ton consumption of the various reagents is noted. Trends in the use of reagents are indicated. Metallurgical results for each class of ore and average screen analysis and alkalinity of the flotation pulps are included.

276. _____ Ore-concentration statistics in 1930. Metallurgical results, flotation reagents, and bibliography. Mineral Resources of the United States, 1930, Bureau of Mines, part I, 1932, pp. 749-791.
Continuation of the above report.

277. _____ Ore-concentration statistics in 1931. Metallurgical results and flotation reagents. Mineral Resources of the United States, 1931, Bureau of Mines, part I, 1933, pp. 417-436.
Continuation of the above report.

278. MOORE, B. H. Western Australian metallurgy. Min. Mag. (London), July 1932, pp. 52-55.
General discussion of the development of flotation as applied to Kalgoorlie ores, with short descriptions of the Lake View & Star and Wiluna mills, both of which use flotation followed by cyanidation of the concentrates. The bromocyanide process used at the Boulder Perseverance mill is also described.

279. MORROW, J. B. The preparation of coal. Canadian Min. and Met. Bull. 231, 1931, pp. 836-854.
General outline and discussion of coal washing, including sizing, mixing, and cleaning.

280. MORTENSON, MAGNE. Activation by copper sulphate in the flotation of blende. Tids. Kjemi Bergvesen, vol. 10, 1930, pp. 118-119.
The time required for the activation of blende by copper sulphate solution depends on the iron content of the mineral and is longest for blende rich in iron (marmatite). The rate of activation also increases with temperature and with the presence of acid in the liquid. The electrical conductivity of blendes varies greatly with the iron content; a pure, iron-free blende from Spitsbergen showed a resistance of 6×10^5 ohms per cubic centimeter, while that of marmatite from Roros containing 15.4 percent iron was only 8×10^{-1} ohm.

281. _____ Graphical analysis of flotation test results. Min. and Met., vol. 12, 1931, p. 496.
An ingenious new method of showing graphically the best conditions for maximum return in the conduct of flotation operations. Contours of equi-extraction and equi-concentration are set up from the test data, and the area included in the highest of both sets of contours represents the optimum condition.

282. MORTENSON, MAGNE. Investigation of economic flotation methods for intimately coalesced ores. Tids. Kjemi Bergveson, vol. 11, 1931, pp. 110-111.

Describes new crushing-flotation flow sheet, which has been tried on a laboratory and semicommercial scale with good results in costs of crushing and extraction. The ore is first crushed to such a degree of fineness that only part of the mineral to be extracted is present as "full grains", the rest as "half grains". The crushed product is subjected to a primary flotation, in which the "full grains" are concentrated to a high-grade product, then the "half grains" are taken out by a secondary flotation, the product of which is subjected to new crushing and further treatment.

283. MOTT, R. C. The Sherritt-Gordon mill. Canadian Min. Jour., vol. 53, 1932, pp. 101-109.

A detailed description of the new flotation mill of Sherritt Gorden Mines (Ltd.) at Sherridon, Manitoba. The ore treated carries copper and zinc but under present operations only the copper is saved. The ore is broken to 4 inches in a jaw crusher located underground and after hoisting in skips is reduced to 1/4-inch in a Symons cone crusher and rolls. Grinding is done in two stages in Hardinge ball mills with rake classifiers. Flotation is done in Fahrenwald machines using soda ash, sodium cyanide, zinc sulphate, thiocarbanilide, Aerofloat, cresylic acid, and pine oil. Recoveries are 95.07 percent of the copper, 75 percent of the gold, and 78 percent of the silver. Flow sheets and operating details are given.

284. MUNRO, A. C., and PEARSE, H. A. Milling methods and costs at the concentrator of the Britannia Mining & Smelting Co. (Ltd.), Britannia Beach, B. C. Inf. Circ. 6619, Bureau of Mines, 1932, 24 pp.

The ore treated at this 7,000-ton concentrator consists chiefly of chalcopyrite and pyrite, with minor quantities of chalcocite and bornite, occurring as replacements in a chlorite schist derived from quartz-diorite porphyry. The sulphide minerals are freed from the hard gangue minerals at 48-mesh but grinding to 200-mesh is necessary to free the sulphide minerals from each other. This condition has resulted in the development of the present flow sheet which requires grinding of the entire feed to about 8 percent plus 48 mesh and making a bulk float of the sulphide minerals followed by regrinding of the concentrates to about 76 percent minus 200-mesh and preferential flotation of the copper and iron minerals into separate concentrates. Primary slime is removed in the crushing circuit before ball milling and treated in a separate flotation circuit. Air-lift flotation machines are used throughout and flotation reagents are lime, pine oil, xanthate, and dicresol-dithiophosphate. About 88 percent of the copper is recovered in concentrates assaying 26.6 percent copper. The article gives details on metallurgical data, cost data, flow sheets, etc.

285. MYNERS, T. F. Magnetic concentration methods and costs of Witherbee, Sherman & Co., Mineville, N. Y. Inf. Cir. 6624, 1932, 27 pp.

Detailed description of equipment and methods used at mills 4 and 5, treating large quantities of magnetite ore. Flow sheets, milling costs, metallurgical data, etc., are given, as well as a brief history of magnetic concentration methods.

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286. **NEEDHAM, S. W.** The collection and treatment of washery slurry. Coll. Guard., vol. 143, 1931, pp. 2044-2046, 2133-2134.
Discusses laboratory experiments on the treatment of coal-washing plant slurry. Various organic and inorganic precipitators were tried in an effort to increase the settling rate of the slurry, making it more amenable to concentration.

287. **NELSON, W. I.** Milling methods and costs at the Superior concentrator of the Engels Copper Mining Co., Plumas County, Calif. Inf. Circ. 6550, Bureau of Mines, 1932, 22 pp.
The concentrator has a maximum capacity of 1,500 tons a day and treats ore from the Engels and Superior mines. Ore contains chalcopyrite and bornite, with very small quantities of pyrite and tetrahedrite. Gangue is a hard quartz monzonite, and the ore requires very fine grinding to liberate sulphides from gangue. Two-stage grinding in ball mills and tube mills is necessary to prepare the flotation feed. Mechanical Sub A machines are used for roughing and Callow pneumatic cells for cleaning and scavenging. Flotation reagents are lime, xanthate, fuel oil, and two kinds of pine oil. Ore treated assays about 1.54 percent copper and concentrates about 29.18 percent copper; recovery is 91.34 percent, with a ratio of concentration of 19.94 to 1. The report gives a history of concentrator operations, details of equipment used, flow sheet, and metallurgical and cost data.

288. **NEVETT, RALPH D.** Changing over from acid to alkaline circuit in flotation practice at the mill of the Zinc Corporation (Ltd.), Broken Hill, New South Wales, Australia. Proc. Australasian Inst. Min. and Met., 1930, pp. 343-353, 445-460.
Detailed description of past and present flotation practice at a mill treating a galena-marmatite ore. Flotation reagents used and metallurgical results are given.

289. **NIEMAN, E. F.** Plant and equipment of Harmony Mines. Min. Cong. Jour., vol. 17, 1931, pp. 98-101, 115.
Describes mine and mill of Harmony Mines Co. near Baker, Idaho. Copper ore is treated by flotation in a plant of 200 tons daily capacity. Flotation reagents are pine oil and P. & E. oil. Recovery of copper is 98.91 percent.

290. **OLDRIGHT, G. L.** Influence of flotation upon smelting. Trans. Electrochem. Soc. (Prepr. 60), 1931, 12 pp.
General effect of flotation upon smelting of copper, zinc, and lead ores has been to supply increasingly richer material and to decrease the amount of lower-grade material that has been treated directly. Smelting practice has been altered to fit changes in nature of concentrates received. In general, smaller amounts of worthless gangue are removed by the smelter per unit of metal received, and the units of plants for treating various gangue constituents have been modified correspondingly or decreased in number per unit of metal made. These changes may be traced back and connected to the removal of various specific minerals in the flotation cell.

291. OLDRIGHT, G. L. Treating a complex ore: Data from experimental work on ores in the Denver laboratories of the Complex Ores Recovery Co. Tech. Paper 499, Bureau of Mines, 1931, 101 pp.
Detailed review of experimental work on ore from the Flin Flon mine in northern Manitoba, including roasting, leaching, and electrolysis of solutions from an ore containing zinc, copper, lead, gold, and silver.

292. O'MALLEY, G. B. Amalgamation and flotation tests on an ore from the Goudreau-Localsh area, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 94-96.
The ore consisted of pyrite carrying gold (15.52 ounces) and silver (1.91 ounces) in a quartz gangue. Amalgamation gives a recovery of about 93 percent of both gold and silver and flotation of 94.2 percent of the gold and 86.3 percent of the silver with 4.0 pounds soda ash, 0.16 pound water-gas tar, 0.3 pound amyl xanthate, and 0.1 pound of pine oil.

293. _____ Concentration of copper ore from Arno Mines (Ltd.), Coxheath, Nova Scotia, Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 82-83.
Massive chalcopyrite with pyrite in a siliceous gangue analyzed 4.89 to 5.54 percent copper, 8.63 to 6.16 percent iron, 8.62 percent sulphur, 77.98 to 61.92 percent insoluble, 0.01 to 0.03 ounce gold, and 0.72 to 0.52 ounce silver perton. Characteristic of flotation results with various reagent combinations were recoveries of 94.4 percent copper, 81 percent gold, and 72.8 percent silver in concentrates assaying 26.4 percent copper, 0.04 ounce gold, and 3.44 ounces silver per ton, with 5.0 pounds soda ash, 0.15 pound cyanide, and 0.1 pound thiocarbanilide per ton in the ball mill and 0.06 pound pine oil per ton in the cell.

294. O'MEARA, R. G. Added recovery by hydraulic sizing of fine material in the land-pebble phosphate district of Florida. Rept. of Investigations 3139, Bureau of Mines, 1931, 5 pp.
Use of hydraulic classification at one plant resulted in an increase of 15 percent in phosphate recovery. At another plant similar equipment raised the grade of the concentrates due to rejection of clay. Hydraulic classifiers in use at three plants have increased the yearly production of phosphate about 81,000 tons.

295. _____ Hydraulic classification raises phosphate washery efficiency. Eng. and Min. Jour., vol. 132, 1931, pp. 529-530.
Describes hydraulic classifiers and includes flow sheets and operating results at three plants.

296. O'MEARA, R. G., and PAMPLIN, J. W. Selective oiling and table concentration of phosphatic sands in the land-pebble district of Florida. Repts. of Investigations 3195, Bureau of Mines, 1932, 6 pp.
The separation of the phosphatic grains from the quartz sand depends on selective oiling of the raw phosphatic sands with $1\frac{1}{2}$ to 2 pounds soap and 4 to 5 pounds fuel oil per ton of feed before table concentration. The phosphate grains agglomerate (quartz is unaffected) and are concentrated by tabling to give a high recovery in a high-grade product. Similar tabling without oiling gave low recoveries in low-grade concentrates.

297. O'MEARA, R. G., GOW, ALEXANDER M., and SCHRENK, W. T. Compendium on limes in hydrometallurgy and flotation. Inf. Circ. 6423, Bureau of Mines, 1931, 54 pp.
Reviews the characteristics, analysis, preparation, and uses of lime in hydrometallurgy and flotation, with particular reference to the behavior of minerals in flotation pulps made alkaline by adding lime. The many functions of lime are discussed, including its action in neutralizing free acids, imparting alkalinity, dispersing slime, coating sulphide minerals, etc.

298. OPPENHEUSER, JULIUS. Ore crushing and sintering plant of the Neunkircher Iron Works. Stahl u. Eisen, vol. 51, 1931, pp. 1165-1167.
Describes a crushing plant with a capacity of 500 tons minette per hour and a Dwight-Lloyd sintering band 2 meters wide with a suction surface of 40 square meters and a capacity of 38 to 46 tons per hour. Gives power-consumption figures for both parts of the plant.

299. OSTWALD, WOLFGANG. The theory of flotation. Kolloid Ztschr., vol. 58, 1932, pp. 179-183.
The simplest connection between three phases of flotation processes is a boundary edge rather than a layer. The simplest case of flotation should be a linear flotation. Film flotation utilizes diphilic collectors, such as oleic acid, which is used in large amount. Linear flotation utilizes triphilic collectors in small amounts and forming monomolecular films. Triphilic collectors contain (1) a group with affinity for metal, (2) a hydrophilic group, and (3) a capillary active group. Examples are xanthates (S:C(OR) (SMe), in which S represents group 1, SMe group 2, and OR group 3), mercaptans, or their alkali-metal derivatives and the Aerofloats. Ethyldixanthogen fails as a reagent, not because of insolubility, but because of the absence of a hydrophilic group. Carbanilide and thiourea, although highly adsorbed by ores, lack capillary active groups. Capillary active groups seldom go higher than the heptyl on account of the insolubility of reagents with longer chains.

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300. PALLANCH, R. A. Milling methods at the Midvale concentrator of the U. S. Smelting, Refining & Mining Co., Midvale, Utah. Inf. Circ. 6492, Bureau of Mines, 1931, 17 pp.
The original milling plant at Midvale was a 90-ton gravity-concentration unit erected in 1905. Additions to the plant raised its capacity to 405 tons by 1915. Jigs and tables were used to make lead concentrates and zinc concentrates, and the pyrite was removed from the dried-zinc product by Huff electrostatic machines. The present straight flotation plant was placed in operation in 1926, treating 750 tons a day. Addition of a sixth grinding mill raised the capacity to its present rate of 1,000 tons a day. Ore treated at the plant consists of crude lead-zinc ore from the U. S. and Lark mines at Bingham and smaller quantities of custom ores from several properties. U. S. crude ore constitutes the major part of the feed and consists of galena, pyrite, and sphalerite (marmatitic), closely associated with quartzite, limestone, porphyry, and talc. Under present operations the ore is reduced to about $\frac{3}{4}$ inch in gyratory and roll crushers, ground to flotation size in two stages using rod mills followed by ball mills (all grinding mills being in closed circuit with spiral classifiers), and treated in Minerals Separation Sub A type cells, making three concentrates (galena, sphalerite, and pyrite) in the order named. Flotation reagents used are sodium sulphite, zinc sulphate, xanthate, coal-tar creosote, and cresylic acid in the lead circuit; copper sulphate, pine oil, coal-tar creosote, xanthate, and hardwood creosote in the zinc circuit; and sodium sulphide in the pyrite circuit. Flotation tailings are passed over pilot tables. The report gives flow sheets, metallurgical and cost data, etc.

301. PARSONS, C. S. Advances made in recent years in the metallurgy of gold including improved processes and equipment. Canadian Dept. Mines, Mines Branch, Mem. Ser. 47, 1931, 8 pp.

302. _____ Concentration of lead-zinc ore from Geneva, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 42-47.
The ore consists of siliceous gangue containing galena, sphalerite and pyrite with low values in gold and silver. It analyzes 3.44 percent lead and 9.41 percent zinc. Excellent results were obtained by flotation. The grade of the concentrates can be varied between wide limits without affecting recoveries of lead and zinc. Gold and silver recoveries fall as the grade of the lead concentrates is raised. Details of tests are given.

303. _____ Concentration of the low-grade banded iron ore from Kaministikwia, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 29-42.
The ore consisted of bands of jasper and hematite (19 percent), mixed with magnetite (24.36 percent), and analyzed 31.0 percent iron and 52.42 percent silica. Magnetic cobbing will not produce a marketable grade of iron ore. Gravity concentration on jigs and tables produced concentrates containing more than 50 percent iron but high in silica (25.73 percent). Its physical properties are good.

304. PARSONS, C. S. Experimental tests on gold-copper ore from the Green-Stabell mine. Canadian Dept. Mines, Mine Branch, Rept. 720, 1931, pp. 116-127
The ore containing pyrite, pyrrhotite, and chalcopyrite in quartz analyzed 0.50 percent copper, 6.80 percent iron, 79.68 percent silica, 0.38 ounce gold, and 0.19 ounce silver per ton. Fine grinding is essential; the finer the grinding the higher the gold recovery. Neither straight flotation nor amalgamation recovers enough gold. Amalgamation followed by flotation gives the best recoveries (80 to 87 percent) and the highest ratio of concentration (21 to 28 into 1). From 92 to 98 percent of the copper is recovered.

305. _____ The metallurgy of gold. Canadian Min. Jour., vol. 52, 1931, pp. 621-624.
Reviews development of the cyanide process and discusses the application of combined flotation-cyanidation processes in treating gold ores.

306. _____ The treatment of gold ores from the English Brook area, Rice Lake district, Manitoba. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 164-169.
A siliceous ore containing a small amount of sulphide and gold, principally native, from three claims, assaying 0.81, 0.77, and 0.28 ounce of gold per ton, gave recoveries of 71.6, 92.2 and 67.9 percent by amalgamation. Cyanidation recovered 96.3, 96.1, and 96.5 percent, with normal consumption of reagents. Flotation concentrates carried 70.7, 85.1, and 71.8 percent of the gold. Amalgamation followed by flotation recovered 97.7, 93.9 and 93.9 percent. Cyanidation of these concentrates gave 92-percent recovery with normal consumption of reagents.

307. PARSONS, C. S., and ANDERSON, A. K. Experimental tests on gold ore from the Perrier mine, Nelson, British Columbia. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 53-58.
The ore was of quartz carrying sulphides of lead, zinc, and iron and 0.29 ounce gold and 0.43 to 0.51 ounce silver per ton. The highest recoveries (gold 91.3 percent, silver 91.9 percent, lead 93.5 percent, and zinc 73.9 percent) were obtained by flotation. Details of other tests using amalgamation, cyanidation, and table concentration are given.

308. _____ Some problems in the control of the flotation process. Canadian Chem. and Met., vol. 15, 1931, pp. 222-223.
States some results obtained by the adding reagents or a combination of reagents or by variations in plant operation. Flotation is a surface-reaction phenomenon, and the theory of interfacial tension is discussed. Plant-scale flotation work showed a lower percentage recovery than laboratory-scale, probably because of oxidation effects. If enough oxygen is present in the water in the ball mill to complete the oxidation of the salts produced to a more or less stable form, a fast-floating condition will result; if not, partly oxidized salts are left in the pulp to interfere with the flotation. By placing a conditioning tank and increasing the time of contact before flotation, less cyanide can be used and a more selective condition obtained between the flotation of the copper and iron or zinc. Water impurities sometimes have an important bearing on flotation.

309. PARSONS, C. S., and ANDERSON, A. K. The concentration of copper-tin ore from the Snowflake Mining Co., Albert Canyon, British Columbia. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 101-116.
An ore of alternate bands of carbonaceous slates, quartz, pyrite, and stamnite analyzed 5.40 percent copper, 4.35 percent tin, 2.76 percent lead, 3.25 percent zinc, 15.3 percent iron, and 33.13 ounces silver per ton. High recoveries (copper 98 percent, tin 98 percent, and silver 95 percent) were made with any of the popular flotation reagents. Lime is detrimental. Soda ash and cyanide are used to depress the pyrite. The grade of the concentrate after one cleaning is 23 percent copper, 19 percent tin, 12 percent lead, and 125 ounces silver per ton.

310. PARSONS, C. S., and GODARD, J. S. Experimental tests on gold ore from the McMillan Gold Mines (Ltd.), Mongowin Township, Sudbury district, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 48-53.
The ore contains pyrite, pyrrhotite, arsenopyrite, and a small quantity of chalcopyrite in a siliceous gangue. It analyzes 10.35 percent iron, 4.73 percent sulphur, 0.87 percent arsenic, 0.04 percent copper, 75.17 percent insoluble, 0.50 ounce gold, and 0.08 ounce silver per ton. Cyanidation extracted 94 to 97 percent of the gold, with reagent consumption of 1.5 pounds cyanide and 4 pounds lime per ton. Flotation followed by tabling recovered 97.1 percent of the gold. Cyanidation of the concentrates gave 92.3 to 97.6 percent of the gold.

311. _____ Flotation tests on copper ore from McGinn claims, Baldwin Township, Sudbury district, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 99-101.
A siliceous ore containing chalcopyrite, pyrite, and pyrrhotite analyzed 2.72 percent copper, 4.35 percent iron, 0.10 percent zinc, and 0.01 ounce gold per ton. Recoveries of 95.7 to 98.5 percent of the copper were obtained in concentrates assaying 11.80 to 21.50 percent copper.

312. PARSONS, C. S., and HALFORD, D. S. The recovery of silver from the tailings of the Castle-Trethewey mill at Bestet, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 96-99.
The tailings assayed 4.50 ounces silver per ton. A high recovery (90 percent) can be made easily at a reagent cost of 10 cents per ton. A high-grade concentrate was not obtained in the small-scale tests.

313. PARSONS, C. S., ANDERSON, A. K., and GODARD, J. S. The recovery of gold from the ore of the Malartic Mines (Ltd.), Amos, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 86-93.
Flotation of a siliceous ore containing 0.23 to 0.39 ounce gold per ton gives a recovery of 77.5 percent of the gold. When preceded by amalgamation a 90.4-percent recovery is made, but there is the disadvantage that 21.4 percent of the gold is in the pyritic concentrates. Cyanidation gave recoveries of 93.8 percent at minus 48-mesh and 95.5 percent at minus 100-mesh in a small-scale test and 96.2 percent in a large-scale test.

314. PARSONS, C. S., GODARD, J. S., and ANDERSON, A. K. The investigation of the recovery of gold and silver in the ore of the Howey Gold Mines (Ltd.). Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 17-29.
The ore consisted of quartz and porphyry containing 2.68 percent iron, 0.04 percent zinc, 0.03 percent lead, 0.71 ounce gold, and 0.32 ounce silver per ton. With minus 48-mesh pulp, 96 percent of the gold and 84 percent of the silver can be obtained with 24 hours agitation in a solution of 2.45 pounds cyanide per ton, with a consumption of 0.10 to 0.30 pound per ton of ore. In 48 hours 97.2 percent of the gold and 84 percent of the silver are recovered. Flotation (28 percent plus 48-mesh) recovers 80 percent of the gold and 81 percent of the silver in a product assaying 6.15 ounces gold and 3.47 ounces silver. Tabling and flotation at minus 40-mesh yields a product assaying 8.56 ounces gold and 2.90 ounces silver per ton and containing 95.4 percent of the gold and 80 percent of the silver. Cyanidation of the concentrates for 72 hours yields 94 percent of the gold and 78 percent of the silver, with a cyanide consumption of 5.8 pounds per ton of concentrate or 0.52 pound per ton of table feed.

315. PARSONS, C. S., WALKER, G. B., and ANDERSON, A. K. The concentration of copper-nickel of the Falconbridge Nickel Mines (Ltd.), Garson, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 29-44.
An ore of sulphides of copper, nickel, and iron in a basic gangue assaying 1.11 percent copper and 2.28 percent nickel was jigged, concentrated on a Butchart table, concentrated on a Plato table, and floated. Gravity concentration gave little improvement. Grinding with 4.0 pounds soda ash per ton of ore, flotation of copper with 0.04 pound cresylic acid followed by flotation with 0.20 pound American Cyanamid Reagent No. 404 or 0.20 pound No. 323 plus 0.20 pound cresylic acid gave recoveries of 99 percent of the copper and 95 percent of the nickel.

316. PARSONS, K. P. W. Milling practice at the Central mine, Broken Hill, New South Wales. Proc. Australasian Inst. Min. and Met., vol. 82, 1931, pp. 23-56.
Detailed description of milling operations at a plant treating a galena-marmatite ore containing 14.5 percent lead and 14.5 percent zinc. Oxidized-lead minerals (8 percent of total lead) cause losses. The flotation reagents used, metallurgical results, and flow sheets are included.

317. PEDDRICK, C. H., JR. Magnetic separation purifies feldspar. Eng. and Min. Jour., vol. 130, 1930, pp. 613-614.
Describes feldspar concentration at the plant of the Tennessee Minerals Products Corporation, Spruce Pine, N. C.

318. PETAR, ALICE V. Molybdenum. Econ. Paper 15, Bureau of Mines, 1932, 38 pp.
Discusses properties, uses, occurrence, mining, milling and manufacturing methods, foreign and domestic production, markets, etc., for molybdenum; includes extensive bibliography.

319. PETERSEN, W. Recent flotation procedures. Kolloid-Ztschr., vol. 58, 1932, pp. 249-255.
Reviews patents covering reconstructed oils, depressors, activators, differential flotation, and nonmetallic flotation.

320. PETERSEN, W. Recent procedures in flotation. Kolloid-Ztschr., vol. 58, 1932, pp. 121-125.
Reviews patents covering flotation reagents.

321. PETERSEN, W., and GERGOR, F. The clarification of coal-washer slimes. Glückauf, vol. 68, 1932, pp. 621-630.
Addition of electrolytes was ineffective. Protective colloids, particularly those of vegetable origin, greatly hastened settling. Swollen potato starch, 100 grams per cubic meter of slime, is an example of an effective colloid for the purpose.

322. PIETERS, H. A. J. The flotation of coal sludges. Brennstoff-Chem., vol. 12, 1931, pp. 325-327.
The amount and ash concentration of foam produced in a specially designed apparatus by the addition of various agents to a coal fines-red clay mixture in water have been determined. Of many agents tested, good flotation was obtained with phenols, alpha-naphthylamine, pyridine, thiocarbanilide, tar oils, turpentine, petroleum, alcohol, mineral oils, and oils easily emulsified as turpentine or cutting oils. Decrease in surface tension is believed responsible for the action of alcohols; reagents such as glycerol, sodium silicate, and alkalies, which increase surface tension, repress foaming. Increase in molecular weights in alcohols is accompanied by increase in flotation power. Viscous oils have very little effect because of the poor covering of the particles. Colloidal solutions, such as saponin and starches, have no effect. Addition of electrolytes, alkalies excepted, had little effect on the foaming agent. Particle size and concentration of the material to be treated are very important.

323. PLAKSIN, I. N., and ZHURLOV, N. L. Recovery of gold and silver from graphitic and nongraphitic waste. Tzvetnuie Metallui, 1931, pp. 1156-1179.
Discusses in detail various methods of recovering gold and silver from graphitic wastes with particular reference to cyanidation.

324. PONCELET, E. F. The Vibro-settler. Canadian Min. Jour., vol. 53, 1932, pp. 302-304.
Describes a V-shaped gravity-concentration device equipped with baffles and spigots and driven by an eccentric.

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325. PRICE, G. S. Milling methods and costs at the Page concentrator of the Federal Mining & Smelting Co., Kellogg, Idaho. Inf. Circ. 6590, Bureau of Mines, 1932, 6 pp.

The Page concentrator is a straight flotation plant, placed in operation in 1926, treating lead-zinc ore intimately associated with pyrite, quartz, siderite, and calcite. The capacity of the mill was increased to 300 tons a day in 1928. A jaw crusher, cone crusher, and a set of rolls are used to reduce the ore to about $\frac{3}{4}$ inch. Ball mills and rake- and drag-type classifiers grind the ore to flotation size in two stages. Minerals Separation machines are used as roughers and Forrester machines as cleaners in both the lead and zinc circuits. Flotation reagents are: Lead circuit--zinc sulphate, Aerofloat, and cresylic acid; zinc circuit--copper sulphate, xanthate, pine oil, and Barrett no. 4. Recoveries are 91.5 percent of the lead, 88.9 percent of the silver, and 35.1 percent of the zinc in lead concentrates assaying 70.8 percent lead; and 55 percent of the zinc in 51.3 percent zinc concentrates from ore assaying 10.95 percent lead, 2.79 percent zinc, and 4.32 ounces silver. Includes flow sheets, mill equipment, metallurgical and cost data, etc.

326. PROBERT, ALAN. Standardized sieving methods. Eng. and Min. Jour., vol. 131, 1931, pp. 311-312.

Describes a sieving procedure designed to give uniform, accurate results and allowing rapid manipulation. The material is wet-screened on a 200-mesh sieve, the parts are dried and weighed, and the oversize is screened on a mechanical shaker in the usual manner. Each sieve is then screened separately to an end point (1 gram of undersize in 1 minute). The results are more reliable than ordinary nest sieving.

327. QUITTKAT, G. Dressing gold ores in Rumania. Metall u. Erz, vol. 29, 1932, pp. 168-174. See also Canadian Min. Jour., vol. 53, pp. 531-536.

General review of gold milling in Rumania, with descriptions and flow sheets of three plants.

328. RALSTON, OLIVER C., and BARKER, LYLE M. Flocculation and froth quality in flotation. Trans. Electrochem. Soc. (Prepr. 60), 1931, 7 pp.

Discusses results of a study of the flocculating effect of various collecting reagents in the treatment of pyritic copper ore from Jerome, Ariz. The conclusions are: (a) The modern soluble-collecting reagents vary between causing the two extremes of "underflocculation" and "overflocculation" of the minerals that pass into the froth, other pulp conditions being constant; (b) by appropriate admixture of collectors of opposite types the degree of flocculation of the minerals entering the froth can be controlled; and finally (c) by such regulation of the mineral flocculation the physical characteristics of the froth can be controlled to the end that improved metallurgical results are achieved. Underflocculated froths look dirty and are voluminous, poorly mineralized, and less selective; overflocculated froths look brilliant and metallic, are overburdened with mineral, and are flat, shallow, and too tough to flow by their own weight or to be raked off easily.

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329. RAMSAY, G. C. The action of cyanide on gold. Chem. Eng. and Min. Rev., vol. 24, 1932, pp. 236-237.

Various gold-cyanide equations are shown, and the potential difference in the solution of gold is determined by a potentiometer, calomel cell, and gold electrode. The effect of various substances, such as oxygen, on the potential difference is determined and the conclusion is that oxygen should aid in the solution of gold in cyanide solutions.

330. READ, B. K. Cyanidation: An historical note. Canadian Chem. and Met., vol. 16, 1932, pp. 30-31.

The failure of salt treatment is pointed out, the MacArthur-Forrest process outlined, and the rise in cyanide production noted.

331. READ, HERBERT N. Milling and cyaniding practice at Teck-Hughes Gold Mines (Ltd.). Canadian Min. Jour., vol. 52, 1931, pp. 839-846.

Describes in detail milling at a 1,000-ton plant at Kirkland Lake, Ontario, giving an historical review of developments. Flow sheets are given, and the mill equipment, cyanide consumption, and many metallurgical details are discussed.

332. REDINGTON, JOHN. Milling methods and costs of the Coniaurum Mines (Ltd.), Schumacher, Ontario. Inf. Circ. 6541, Bureau of Mines, 1931, 5 pp.

Considers operations at the 500-ton all-slime cyanidation mill, Schumacher, Ontario, treating $6.00 gold ore in which the gold is associated closely with minor quantities of sulphide minerals in a quartz and schist gangue. A gyratory crusher and two sets of rolls are used to reduce the ore to about ½ inch. Tube mills, in closed circuit with rake classifiers grinding in cyanide solution, prepare the pulp for agitation. After two stages of agitation the pulp flows to a bowl classifier, which removes coarse sand and returns it to the tube mills. This finishing classifier has a decided concentrating action on the sulphide minerals, providing additional grinding and much longer cyanide contact. Thickeners and two-stage filtration complete the pulp-handling equipment. The usual clarifying, precipitating, and refining equipment is provided.

333. REEDER, E. C. Milling methods and costs at the Hillside Fluorspar Mines, Rosiclare, Ill. Inf. Circ. 6621, 1932, 20 pp.

Describes jig and table concentration of an ore containing fluorite, calcite, limestone, sandstone, chert, and a little galena. Marketable concentrates of fluorspar and galena are made. Separation by gravity methods is difficult due to the slight difference in specific gravity of fluorite and calcite, and careful sizing of jig and table feed is necessary.

334. RICHARDS, ROBERT H., and LOCKE, CHARLES E. Progress in ore dressing and coal preparation in 1930. Mineral Ind., vol. 39, 1930, pp. 671-725.

Detailed review of ore-dressing developments during 1930, including discussions of new equipment and processes.

335. _____ Progress in ore dressing and coal preparation in 1931. Mineral Ind., vol. 40, 1931, pp. 613-675.

Similar review for 1931.

336. RICHARDSON, A. C., and GANDRUD, B. W. Re-treatment of Sayreton jig middlings on coal-washing tables. Rept. of Investigations 3101, Bureau of Mines, 1931, 6 pp.
The Sayreton middlings contain enough low-ash coal to warrant re-treatment. Virtually all is found in material finer than 3/16 inch, which is ideally suited for tabling. A test reduced the minus 3/16-inch jig middlings containing 21.3 percent ash to 9.5 percent ash, with a recovery of 73.8 percent. Sulphur was reduced from 1.00 to 0.74 percent.

337. RICHARDSON, A. C., COE, C. D., and ANTHONY, H. L. Washability studies of the Brookwood bed at the Warrior View mine, Tuscaloosa, Ala. Rept. of Investigations 3170, Bureau of Mines, 1932, 14 pp.
It is difficult to reduce the ash below 3 percent. Crushing the 3-inch to ¾-inch coal to pass 3/16-inch would make possible an additional ash reduction of about 1 percent.

338. ROACH, A. C. Magnetic concentration. Canadian Min. Jour., vol. 52, 1931, pp. 551-553.
Discusses application of magnetic concentration in treating Sudbury copper-nickel ores. The process may be applied as a scalping operation preceding table or flotation concentration.

339. ROBINSON, BERTRAND. The Granada gold mine. Canadian Min. Jour., vol. 53, 1932, pp. 53-57.
Describes 150-ton blanket-concentration mill treating $12 to $15 gold ore from the Granada mine near Rouyn, Quebec. The plant was originally designed as an amalgamation plant, but fouling of the plates necessitated the adoption of blankets.

340. ROLLER, PAUL S. Separation and size distribution of microscopic particles. An air analysis for fine powders. Tech. Paper 490, Bureau of Mines, 1931, 46 pp.
Discusses development of an apparatus using air for exact separation of particle sizes in fine powders. Gives results of tests.

341. ROTHELIUS, ERNST. The iron losses in magnetic separation I. Jernkontorets Ann., vol. 116, 1932, pp. 1-19.
Considers metallurgical losses at four mills using magnetic separators and gravity-concentration equipment.

342. ROWE, R. C. Grinding Buckingham feldspar. Canadian Min. Jour., vol. 52, 1931, pp. 613-615.
Briefly describes feldspar-grinding plant of the Canadian Flint & Spar Co. (Ltd.).

343. _____ The new mill of Aldermac Mines (Ltd.). Canadian Min. Jour., vol. 53, 1932, pp. 149-153.
Describes flotation plant near Noranda treating a massive pyrite-chalcopyrite ore assaying 40 percent iron and 2 percent copper. Soda ash, cyanide, thiocarbanilide, pine oil, and xanthate are used to make 25-percent copper concentrates recovering 90 percent of the copper. Water-gas tar and xanthate are used in the pyrite circuit to make iron concentrates. The quantity of cyanide used is the most important milling factor.

344. RUTH, B. F., MONTONNA, R. E., and MONTILLON, G. H. Comments upon recent developments in the theory of filtration. Ind. Eng. Chem., vol. 23, 1931, pp. 850-851.
Equations, based on Poiseuille's law, of general applicability for filtration at constant pressure and at constant rate were derived from experimental data. Filtration tests were made on Sil-O-Cel, calcium carbonate, calcium sulphate, ferric hydroxide, and lead chromate, with cotton cloth and Monel-metal cloth and Filtros-plate filter septums. For slightly compressible material the constant-rate method was superior to the constant-pressure or constant-resistance methods.

345. SCOTT, MATTHEW. The treatment of Hollinger precipitate to produce fine gold. Canadian Min. Met. Bull. 228, 1931, pp. 466-476. See also Bull. Inst. Min. Met., vol. 318, 1931, 10 pp.
Describes refining of gold and silver at the Hollinger mill. The zinc-dust precipitate is treated first in hydrochloric acid and then in sulphuric acid before the gold is melted into bars.

346. SCOTT, W. G. Slime-leaching plant at Inspiration. Arizona Min. Jour., vol. 14, 1931, pp. 7-11.
Discusses removal of primary slime from the main leaching-plant feed and treatment of the slime by flotation and acid leaching at the plants of the Inspiration Consolidated Copper Co., Inspiration, Ariz. Higher copper recoveries and improved operations at the main plant have resulted from this change.

347. SCOTT, W. P. Milling methods and ore-treatment equipment (at Alaska Juneau). Eng. and Min. Jour., vol. 133, 1932, pp. 475-482.
Describes operations at the Alaska Juneau mill, which treats 11,000 to 13,000 tons of gold ore a day. Significant features are stage crushing, with two stages of hand sorting, which eliminates as waste about 45 percent of the total feed, use of concentrating tables in three stages, and flotation of only a minor part of the total feed.

348. SHEPARD, O. C. Bubble attachment in flotation. Min. and Met., vol. 13, 1932, pp. 282-283.
Mathematical analysis of bubble attachment to mineral particles, in which an equation is established by which the force of attachment can be measured. This force is an exact measure of floatability.

349. SIEDLER, PHILIPP. Xanthates as collectors. Metall u. Erz, vol. 28, 1931, pp. 425-429.
Studies mechanism of action of xanthates in flotation. The collecting action is due to adsorption of the xanthate ion itself, rather than to coalescence of the products of hydrolysis.

350. SIEDLER, PHILIPP, MOELLER, A., and REDDEHASE, TH. The theory of flotation. Kolloid-Ztschr., vol. 60, 1932, pp. 318-324.
The Ostwald theory does not agree with experimental facts. Photomicrographs show that galena surfaces are in contact with the gas bubble-liquid interface and that the particles do not extend into the bubble. Ethyldixanthogen and thiocarbanilide are effective reagents when added in alkaline solution. Their lack of effect in ordinary flotation is due to insolubility and not to the absence of one of Ostwald's triphilic groups. Collectors must have 2 groups, a metallophilic and an aerophilic.

351. SMITH, E. H. Cyanidation of a copper-gold ore. Canadian Chem. and Met., vol. 15, 1931, pp. 45-46.
The function of lime in cyanidation of gold ores is attracting much attention. Lime and alkalinity have no solvent action of the precious metals but enter into numerous reactions occurring in the process. A pyritic ore ground in lime solution yields calcium sulphide and calcium sulphite, and gangue constituents consume a certain amount of lime. Each mineral in an ore removes lime from the mill solution in an amount that depends on the lime present, and this amount is specific for each mineral. Lime decomposes the zinc cyanide or double cyanide formed in the precipitation processes.

352. SMITH, E. S. ROMILLY. The gold deposits of Nova Scotia: The cause of unprofitable operations and the remedy. Canadian Min. and Met. Bull. 242, 1932, pp. 308-316.
Only 50 percent of the gold content of Nova Scotia ore is recovered by amalgamation; the balance is in sulphides. Analyses of concentrates of tailings from 20 mills (after extraction of free gold) show 21 to 40 percent metallic arsenic and $10 to $60 of gold. A flow sheet is suggested.

353. SOMER, OSKAR. Flotation of kaolin. A study of the flotation of oxidized minerals. Metall u. Erz, vol. 29, 1932, pp. 265-268.
Discusses direct and indirect flotation of oxidized minerals.

354. _____ Nonmetallic mineral flotation. Eng. and Min. Jour., vol. 133, 1932, pp. 568-569.
Considers flotation of kaolin from quartz and mica. No selective wetting could be accomplished, but with all minerals wetted with sodium silicate selective coagulation of the kaolin could be effected by using aluminum chloride.

355. STEE, C. O. Amalgamating and cyaniding on Siscoe Island, Quebec. Eng. and Min. Jour., vol. 133, 1932, pp. 319-322.
Describes 180-ton mill treating gold ore from the property of Siscoe Gold Mines (Ltd.), on Siscoe Island, Quebec. A small hydraulic classifier on the ball mill discharge recovers about 78 percent of the gold, and blankets recover 7 percent additional. Cyanidation recovers 13 percent of the total gold. Over-all recovery is about 98 percent on $15.45 ore. Mill flow sheet is included.

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356. STEIN, OTTO. Calculation of metal yield in ore-dressing processes. Metall u. Erz, vol. 28, 1931, pp. 140-143.
Discusses methods of calculating yields in the various ore-dressing processes.

357. STROUP, R. J. Eagle Picher's central milling plant. Eng. and Min. Jour., vol. 133, 1932, pp. 289-292.
Construction was started in April 1932 of a 3,600-ton centrally located milling plant by the Eagle Picher Mining & Smelting Co. to treat zinc and lead-zinc ore from 18 properties in Oklahoma and 6 in Kansas. Proposed flow sheet is included.

358. SULLIVAN, JOHN D. Chemistry of leaching bornite. Tech. Paper 486, Bureau of Mines, 1931, 20 pp.
Bornite can be leached with ferric sulphate. The rate of solution was virtually independent of the strength of solvent if enough reagent was present. As ferric salts reduce current efficiency and necessitate higher current densities in electrolytic precipitation it is desirable to keep their concentrations as low as possible. If the ferric iron remained constant the rate of solution was independent of acid strength (0.25 to 10 percent). The rate for 10- to 28-mesh particles was virtually identical with that for 100- to 200-mesh mineral. Smaller material was attacked only slightly more rapidly and larger material only slightly more slowly. Temperature had a marked effect. Leached particles retained practically their original form. The final particle was composed of sulphur and insoluble material. Sulphuric acid and air will attack bornite more slowly than ferric sulphate. Ferric chloride was more effective than ferric sulphate.

359. _____ Chemistry of leaching covellite. Tech. Paper 487, Bureau of Mines, 1931, 18 pp.
Covellite can be leached with ferric sulphate. The rate of solution was virtually the same for minus 100- plus 200-mesh covellite in solutions containing 1 and 2 percent of ferric iron as for minus 10- plus 28-mesh material. The rate was practically independent of acid strength for the first 8 days, when the ferric iron concentration remained constant at 1 percent. Sulphuric acid in the presence of air attacked covellite at about half the rate of ferric sulphate. Increased temperature increased the solution rate markedly. Artificial covellite produced by leaching chalcocite dissolved at a much faster rate than natural covellite.

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360. SULLIVAN, JOHN D. Factors involved in the heap leaching of copper ores. Inf. Circ. 6425, Bureau of Mines, 1931, 12 pp.
Enough solution cannot enter a rock at one time to dissolve the copper minerals completely unless the copper content is very low. Frequent renewals of solution and much time are required. In heap leaching with oxidized ores the slowest step is probably removal of the soluble salts. If the copper exists as sulphide (chalcopyrite excepted) about 20 days will be required for complete dissolution of the copper. The rate of dissolution of any copper mineral by a given solvent cannot be changed markedly. Displacing air in heaps with sulphur dioxide from waste smelter gases increases the rate of solution penetration. Finer crushing or recirculation of solutions (if containing some free acid) speeds extraction.

361. SULLIVAN, JOHN D., and BAYARD, K. O. Extraction of soluble copper from ores in leaching by percolation. Rept. of Investigations 3073, Bureau of Mines, 1931, 43 pp.
The physical nature of the rock, its porosity, the degree of kaolinization and sericitization, and the amount of natural slime determine the amount of solution that will be retained and the difficulty that will be found in washing out the soluble salts. The rate of removal of soluble copper decreased with increases in the amount of slime present and increased only slightly with temperature. Wet charging of the ore into the vat was advantageous. In washing, the cycles of submergence and draining should be as close to each other as possible, consistent with fairly thorough mixing of the copper sulphate in the wash solution and thorough draining of the solution. The copper sulphate concentration was greater at the bottom than at the top of the ore column. Circulating the solutions increased extraction rates. Piston-displacement washing removed copper more rapidly than batch washing, but a greater volume of wash solution was required. Downward and upward percolation of wash solutions gave virtually identical extraction rates. Less copper sulphate was precipitated in the ore when the impregnating solutions were acidic.

362. SULLIVAN, JOHN D., and OLDRIGHT, G. L. Leaching oxidized copper ores: Effect of strength of acid in leaching solvent. Rept. of Investigations 3106, Bureau of Mines, 1931, 9 pp.
Gives results of test leaching of Ajo and Silverbell oxidized copper ores. Increasing acid strength of solvent increases solution rate but not in proportion. Acid consumption increases with lime and higher concentration.

363. SULLIVAN, JOHN D., and OSTREA, E. O. Factors governing the entry of solutions into ores during leaching. II. Tech. Paper 498, Bureau of Mines, 1931, 23 pp.
Solutions penetrate into small-bore or capillary glass tubes more rapidly when the tubes are filled with sulphur dioxide than with air. Replacing the air in the voids within pieces of Warren unweathered ore with sulphur dioxide increased the rate of solution penetration.

364. SULLIVAN, JOHN D., and TOWNE, A. P. Agglomeration and leaching of slimes and other finely divided ores. Bull. 329, Bureau of Mines, 1931, 60 pp.
Copper in concentrator tailings can be recovered by agglomerating the ore and leaching by the open-drainage trickle method. Coarse material must be added to the sand and slimes to act as nucleus upon which the fine material can build. About 10 percent moisture is necessary. Solutions percolated freely through agglomerated ore containing up to 30 percent minus 200-mesh material, with 90 percent extraction of acid-soluble copper and 80 percent of total copper when the ratio of oxidized to sulphide copper was approximately 1 to 1. By recirculation, copper concentration was built up to 61 grams per liter. Experiments were conducted on all scales up to 12 tons.

365. _____ Leaching copper ores: Advantages of wet charging. Rept. of Investigations 3050, Bureau of Mines, 1931, 26 pp.
Charging the ore into the leaching vat in the wet condition increases extraction of copper. Virtually identical extraction was obtained by upward and downward percolation when the solutions were not continuously circulated. In the latter case, extraction was greater on downward percolation. Virtually idential extraction resulted from flood and trickle leaching when the ore was wet-charged. With dry-charged ore extraction was greater from plus 100-mesh ore than from unsized material containing fines; there was no difference when the ore was wet-charged.

366. SUNIER, ARTHUR A., and WEINER, LAW G. Solubility of gold in mercury. III. Jour. Am. Chem. Soc., vol. 53, 1931, pp. 1714-1721.
Results of about 50 determinations of the solubility of gold in mercury at temperatures ranging from 200 to 300°.

367. SWAINSON, S. G. Progress in flotation and equipment. Eng. and Min. Jour., vol. 131, 1931, pp. 69-71.
A general review of developments in flotation. Discusses flotation of manganese, phosphate rock, fluorspar, barite, coal, bauxite, mica, calcite, kaolinite, feldspar, lead carbonate, native copper, gold and silver ores, and sulphide ores. Briefly summarizes new developments in flotation equipment.

368. SWAINSON, S. G., and ANDERSON, A. E. The promoter activity of alkyl xanthates. Trans. Electrochem. Soc. (Prepr. 60), 1931, 12 pp.
Alkyl xanthates have been prepared from a number of normal, iso-, and secondary alcohols containing 2 to 7 carbon atoms. The relative promoter activity of these xanthates has been determined by flotation tests on typical copper and lead ores. The promoter activity attains a maximum at 4 to 5 carbon atoms in the alcohol. The promoter activity of the branched-chain-alcohol xanthates is greater than that of the corresponding straight-chain-alcohol xanthates.

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369. TAGGERT, ARTHUR F. Mineral flotation. Jour. Phys. Chem., vol. 36, 1932, pp. 130-153.
Detailed theoretical consideration of flotation reactions. Basic principles are explained and flotation processes divided into two groups, (1) bubble-column processes and (2) pulp-body processes, giving examples of each class. The functions of various reagents, such as frothers and collectors, are discussed in detail.

370. TAKAKUWA, TAKESHI. Some surface-chemical studies contributing to the theory of flotation. Proc. World Eng. Cong., Tokyo, 1929, vol. 35, 1931, pp. 395-410.
Ascertains intimate relationship between coagulation and flotation by parallel experiments of flotation and change of settling velocity. Solid particles which are amenable to flotation cannot be floated until the film of oleic acid around them reaches about 2 molecules in thickness. An electrolyte of a complicated molecular form, such as a double salt, is very destructive, and that of a simple one has no influence. A possible mechanism of flotation is described.

371. TAKATA, MINORU. Size of flotation feed and floatability of minerals. Proc. World Eng. Cong., Tokyo, 1929, vol. 35, 1931, pp. 385-394.
Discusses effect of grain size on flotation and presents a formula for determining the maximum grain size of a floatable mineral particle. Treats effect of mineral hardness.

372. TALMUD, D. L. Mechanism of the flotation process. Physik. Ztschr. Sowjetunion, vol. 1, 1932, pp. 603-618.
Briefly reviews subject, including mechanical strength of adsorbed films of various degrees of saturation, formation of the flotation arch, the role played by the isoelectric point in making a selection between two types of particles, and the effect of cinematographic study of the mechanism of rupture of bubble films, both protected and unprotected by adsorbed films of soap. Includes mathematical and experimental study of carrying capacity of adsorbed films. The experimental results are much higher than the theoretical. This additional strength is explained by the formation of a flotation arch made up of froth and the solid particles floated upon it.

373. TALMUD, D. L., and SUKHOVOLSKAGA, S. D. Stability of primary foam. Ztschr. physikal. Chem., Abt. A, vol. 154, 1931, pp. 277-308.
Insoluble films of surface-active substances stabilize primary foam, prior to the formation of a saturated adsorption layer. Such foams, and also those of solutions containing dissolved capillary-active substances, show lyophilic properties toward electrolytes. The stability of primary foam formed in electrolyte solutions increases with the concentration of surface-inactive electrolyte. The action of evaporation on stability was examined. A theory of foam stability is proposed. Stability measurements were carried out on many combinations of substances.

374. TEUFER, G. Operating practice in the copper-flotation plant of Minas do Valle do Vouga, Portugal. Metall u. Erz, vol. 28, 1931, pp. 125-128.
Describes operations at flotation plant treating copper-zinc-iron ore.

375. THOMPSON, M. R. Cyanides in metallurgy. Trans. Electrochem. Soc. (Prepr. 60), 1931, 11 pp.
Reviews status of the production and metallurgical uses of cyanides, showing that thousands of tons of this commodity are used annually in electroplating metals, extracting gold and silver from ores, and case-hardening ferrous materials. Discusses advantages and disadvantages of cyanides in detail. Mentions some problems on which research is needed. The data given refer to the United States in particular.

376. THURLOW, J. R. A research on location and constitution of residual gold after cyaniding. Jour. Chem. Met. and Min. Soc. South Africa, vol. 32, 1932, pp. 311-322.
After intensive recyanidation of current cyanide-plant tailings most of the gold remaining in the final residues was found to be associated with the pyrite, largely in the coarser sizes. Selective grinding to 200-mesh would improve extraction.

377. TRAILL, R. J. The separation of copper from molybdenum in molybdenite concentrates by hydrometallurgical methods. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 186-187.
An ore containing chalcopyrite and molybdenite in a siliceous gangue assaying 4.6 percent copper and 2.8 percent molybdenum gave a recovery of 72 percent of the molybdenum by flotation in 50-percent molybdenum concentrates carrying 3 to 5 percent copper and 93 percent of the copper in 25-percent copper concentrates. Roasting the molybdenum concentrates with excess sulphur in a nonoxidizing atmosphere, followed by leaching with ferric chloride, gave 85-percent copper extraction and left 0.5 percent copper with the molybdenite. Roasting at various temperatures and leaching with soda and other solvents gave unsatisfactory results because of the sulphates formed from the alkaline earths, which precipitate with the molybdenum as calcium sulphate when lime is added to the solution to precipitate the calcium molybdate. Removal of the alkali earths with hydrochloric acid before roasting gave molybdenum recoveries of 80 to 90 percent and copper returns of 90 percent. Costs are high.

378. TRAILL, R. J., McCLELLAND, W. R., and JOHNSTON, J. D. Preliminary tests on treatment of titaniferous magnetite from Mine Centre for recovery of iron, titanium, and vanadium. Canadian Dept. Mines, Mines Branch, Rept. 720, 1931, pp. 178-185.
Detailed results of experimental work on ore containing iron, titanium, and vanadium. Concentration followed by metallization and leaching with ferric chloride recovered most of the iron. Vanadium is recovered from the residue by roasting and alkali leaching. Titanium can be recovered by the sulphate process.

379. TRUSCOTT, S. J., and SULMAN, H. L. Final report of I. M. M. representatives on the sieves and screen committee of the British Engineering Standards Association. Bull. Inst. Min. and Met., vol. 311, 1930, 9 pp.
The proposed fine testing sieves are based on a ratio of the fourth root of 2 between linear aperture measurements of consecutive sieves. The American and proposed British sieves will be essentially identical as to aperture but not as to mesh because of the difference in American and British wire gage.

380. TUCKER, L. E., and SWAINSON, S. J. Developments in flotation-cyanidation. Eng. and Min. Jour., vol. 133, 1932, pp. 618-620.
Abstract of Technical Paper 20, American Cyanamid Co. Reviews development of flotation in the treatment of gold ores is given. Gold ores are divided into 6 groups, and 10 combinations of processes are listed for the treatment of gold ores. The growing importance of flotation-cyanidation is pointed out, citing operating examples. Discusses reagents, equipment, and conditions.

381. TURNER, G. W. EASTON. The Ashanti Goldfields Corporation. Min. Mag. (London), vol. 46, 1932, pp. 329-337.
History and description of present operations at the Ashanti property in Gold Coast Colony, Africa. The milling plant treats 100,000 to 142,000 tons of gold ore yearly by roasting and cyanidation. The presence of graphite and gold-bearing arsenic minerals necessitates fine dry grinding and careful roasting. Gives flow sheets and metallurgical data.

382. TURNER, J. R. Progress at the Jack Waite. Arizona Min. Jour., vol. 14, 1931, p. 42.
A brief description of the new 500-ton flotation mill at the Jack Waite mine, Shoshone County, Idaho. Includes flow sheets and description of the equipment used.

383. _____ The Jack Waite Mine. Min. Cong. Jour., vol. 17, 1931, pp. 671-673.
Describes mining and milling at property in northern Idaho.

384. UNGER, F. A., EWING, S. E. T., and WILLEY, J. L. Daggafontein Mines, parts I and II. Min. Mag. (London), 1932, pp. 265-279.
Detailed article on history, geology, development, mining methods, and treatment plant of Daggafontein Mines (Ltd.), operating a large property on the Far East Rand. The plant treats 50,000 tons of gold ore a month by all-slime cyanidation, using three-stage tube-mill grinding in closed circuit with bowl classifiers. Flow sheets and metallurgical data are given.

385. URAZOV, G. G., and ROMANOV, M. M. Hydrometallurgical treatment of Ural nickel ores. Tzvetnuie Metallui, vol. 5, 1930, pp. 1136-1144.
Describes a process of recovering nickel from oxidized ores by using sulphur dioxide (gas or solution) in the presence of ferrous sulphate.

386. VANDERBURG, W. O. Methods and costs of concentrating tungsten ores at Atolia, San Bernardino County, Calif. Inf. Circ. 6532, Bureau of Mines, 1931, 12 pp.

The Atolia Mining Co. operates two milling plants, one of 500 tons daily capacity treating tungsten gravels from placer-mining operations and the other of 100 tons capacity treating scheelite ore from lode-mine operations. Both plants employ jig and table concentration, followed by drying and magnetic separation to remove magnetite. Concentrates are sacked for shipment. Placer gravels are broken to 2 inches in a disintegrating trommel and fed to Harz jigs. Jig concentrates are finished; the hutch product is passed over tables to clean it and dried and treated in the Wetherill machine. Lode-mine ore is reduced to 1/4-inch in a jaw crusher and cone crusher and treated in Harz jigs, the jig hutch product being tabled.

387. _____ Methods and costs of concentrating scheelite ore at the Silver Dike mill, Mineral County, Nev. Inf. Circ. 6604, Bureau of Mines, 1932, 12 pp.

Describes gravity concentration of a tungsten ore at a 45-ton mill near Mina, Nev. The ore averages about 1 percent scheelite, and the gangue is chiefly quartz. The ore is crushed in two stages by jaw crushers and in three stages in rolls, sized by hydraulic classifiers, and treated on tables. Table middlings are returned to the roll circuit, and the concentrates are dried, roasted, and treated by magnetic separation to remove iron. The finished concentrates, assaying 65 to 70 percent tungsten trioxide, are sacked for shipment. Recovery averages about 80 percent. Experimental work (Rolla, Mo.) indicated that fine grinding and flotation would recover 90 to 96 percent of the tungsten, using pine oil, oleic adic, sodium oleate, sodium carbonate, and sodium silicate as reagents.

388. _____ Milling methods at the Hughesville concentrator of the St. Joseph Lead Co., Hughesville, Mont. Inf. Circ. 6447, Bureau of Mines, 1931, 15 pp.

The Hughesville mill, a straight flotation plant of 400 tons capacity, was built in 1928 to treat ore containing galena, sphalerite (marmatitic), and pyrite, with a little chalcopyrite, occurring in an altered syenite and rhyolite gangue. The ore is crushed to about ½-inch in jaw and cone crushers at the mine and transported about 2 miles in an aerial tramway to the mill. Grinding is done in rod mills in closed circuit with rake classifiers. Fahrenwald flotation machines are used as roughers in both the lead and zinc circuits, and St. Joe cleaners and scavengers are also used in both circuits. Lead-circuit reagents are soda ash, cyanide, sodium sulphite, cresylic acid, and xanthate. In the zinc circuit lime, copper sulphate, pine oil, and xanthate are added. Ore treated in 1929 averaged 6.21 percent lead, 5.01 percent zinc, and 9.09 ounces silver. Lead concentrates averaged 61.06 percent lead, 5.71 percent zinc, and 50.19 ounces silver, a recovery of 95.46 percent of the lead. Zinc concentrates averaged 1.17 percent lead, 51.20 percent zinc, and 30.86 ounces silver, a recovery of 80.76 percent of the zinc. The report gives flow sheets, metallurgical and cost data, etc.

389. VARY, RONALD A. Amalgamation practice at Porcupine United Gold Mines (Ltd.), Timmins, Ontario. Inf. Circ. 6433, Bureau of Mines, 1931, 5 pp. (See also Eng. and Min. Jour., vol. 131, 1931, p. 463).

A short paper describing crushing, grinding, amalgamating, and blanket concentration at a mill treating 25 tons of gold ore a day. The paper is designed to show what may be expected in costs at a small mill treating ore from a mine in the development stage. Total milling costs are $1.85.

390. VENDENSKY, D. N., and DUSCHAK, L. H. Starch in the flotation of gold ores. Eng. and Min. Jour., vol. 133, 1932, pp. 334-335.

Describes use of soluble starch as a depressant for clay and talc in the flotation of gold ores. True soluble starch is necessary, and a method of preparation is given. Starch also will depress gold and sulphides, and copper sulphate is used as a selective activator. Recoveries were about the same, but the grade of concentrates was greatly improved. Soluble starch may find application in the flotation treatment of other ores.

391. VON COLLANI, G. Gold in the State of Minas Geraes, Brazil. Its occurrence and recovery. Metall u. Erz, vol. 28, 1931, pp. 523-528.

Discusses geology and metallurgy of the gold deposits in Minas Geraes.

392. VON SCHON, RICHARD. Flotation of graphite. Metall u. Erz, vol. 29, 1932, pp. 181-182.

Amorphous earthy graphite cannot be concentrated by flotation, but the crystalline variety is easily floated. Methods are described.

393. WALKER, G. B. Concentration tests on ore sample from Sudbury Offsets (Ltd.), Levack, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 44-50.

A norite gangue with chalcopyrite and pyrrhotite and assaying 0.78 percent copper, 2.43 percent nickel, 26.10 percent sulphur, and 24.21 percent iron was concentrated by various methods. Tabling, followed by flotation, gave no marked improvement. Flotation, followed by tabling, gave high recoveries of values in low-grade concentrates. Flotation in a circuit made alkaline with soda ash did not produce high-grade concentrates. Use of lime gave high-grade concentrates with fair recovery. Cyanide partly depressed nickel and tended to retard copper flotation if excessive amounts were used. Nickel-iron flotation could be effected by the use of sodium sulphide, copper sulphate, or sulphuric acid. A powerful promoting reagent, such as xanthate or Reagent no. 323, must be used.

394. _____ Flotation tests on ore sample from Amulet Mines (Ltd.), Rouyn, Quebec. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 94-101.

Complex ore of chalcopyrite, sphalerite, and pyrite assaying 3.95 percent copper and 20.52 percent zinc was treated by flotation. Use of Reagent no. 301 in conjunction with thiocarbanilide will effect higher recovery of copper consistent with concentrate grade than the combination of thiocarbanilide and xanthate. Increased amounts of Aerofloat no. 25 resulted in lower zinc tailings. Sodium Aerofloat effected production of the highest-grade zinc concentrates, an average of 56.63 percent zinc. Reagent no. 208 gave higher-grade zinc concentrates than xanthate.

395. WALKER, G. B. Flotation tests on ore sample from Denison Copper Mines (Ltd.), Worthington, Ontario. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 50-54.

Chalcopyrite-pyrrhotite-norite ore assaying 1.27 percent copper, 1.15 percent nickel, and 5.73 percent sulphur was treated by flotation. In a circuit made alkaline with soda ash, bulk copper-nickel concentrates can be obtained with cresylic acid. High recovery of copper is indicated. The addition of 0.10 pound per ton of Aerofloat no. 25 increases recovery of copper and nickel in a lower-grade product. For the flotation of a nickel-iron product, a water-soluble promoting agent must be used. In a lime circuit copper concentrates of much higher grade can be produced. The use of cyanide in conjunction with lime has resulted in the most marked separation obtained. After the nickel-bearing pyrrhotite has been depressed by lime and cyanide it is most difficult to float.

396. ______ Flotation tests on ore sample from Sherritt-Gordon Mines (Ltd.), Sherridon, Manitoba. Canadian Dept. Mines, Mines Branch, Rept. 724, 1932, pp. 18-24.

An ore assaying 2.53 percent copper (chalcopyrite), 4.98 percent zinc (sphalerite), 38.65 percent iron, 21.50 percent insoluble, 0.01 ounce gold, and 0.51 ounce silver per ton gave copper recoveries by flotation of 90 to 94 percent in 12 to 15 percent rougher concentrates, depending on the reagent used. Flotation of the sphalerite in a high-grade concentrate presents considerable difficulty because of the fast-floating pyrrhotite.

397. WALLSOM, H. E. Modern practice in coal cleaning. Fuel Econ., vol. 7, 1932, pp. 379-384. Part II. Trough washers: The Rheolaveur washer, pp. 431-435; part III. Jig washers: Humboldt and Baum types, pp. 485-489.

398. ______ The theory of coal cleaning. Fuel Econ., vol. 7, 1932, pp. 211-217.

Discusses occurrence of ash in coal and its influence on the combustion of coal. Gives method of constructing "washability" curves, and reviews laws regulating the motion of particles in water. Points out practical deviations from these laws.

399. WARK, IAN W., Electrostatic explanation of the phenomenon of flotation. Nature, vol. 129, 1932, p. 833.

The potential of the surface decreases, even when the emulsion is so dilute that its character is lost. If this means a transition to true solution, adsorption from solution may be accompanied by a decrease in the potential of the surface, and negative ions may be adsorbed, not positive ions, as stated by Kamienski.

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400. WARK, IAN W. The physical chemistry of flotation. Trans. Electrochem. Soc. (Prepr. 61), 1932, 8 pp.
More attention should be paid to the study of the contact between mineral and air. Data so far published are very confusing. The driving force of the displacement of water from the surface cannot be expressed in terms of "heat of wetting" alone. With sphalerite, experiments support the view that the function of cyanide is to prevent activation of, or to deactivate, the sphalerite. There is a very decided divergence between peptization and floatability in cyanide solutions of high concentration. There appears to be a definite connection between flocculation and flotation, but whether the phenomena are mutually interdependent or whether each depends upon some more fundamental variable is yet to be decided. Experiments indicate that air does not displace water from the surfaces of the heavy metal sulphide minerals unless they are modified by some reagent.

401. WARK, IAN W., and COX, ALWYN B. Principles of flotation: An experimental study of the effect of xanthates on contact angles at mineral surfaces. Tech. Pub. 461, Am. Inst. Min. and Met. Eng., 1932, 48 pp.
Detailed preliminary report of a special investigation into the fundamental principles of flotation. Describes improved apparatus for measuring the contact angle between mineral and air bubble and discusses effect of various xanthates on this angle with various minerals.

402. WARTENWEILER, F. A review of deaeration of cyanide solution. Jour. Chem. Met. and Min. Soc. South Africa, vol. 32, 1932, pp. 216-218.
A general discussion of the need and methods of deaerating cyanide solutions.

403. _____ The use of a composite grinding load in tube milling on the Witwatersrand. Jour. Chem. Met. and Min. Soc. South Africa, vol. 32, 1932, pp. 262-282.
Considers the use of mixed steel balls and ore pebbles (about 1 or 2) in tube-mill grinding. Power consumption is higher, but capacities and grinding are better resulting in higher extractions. No increase in total costs was noted in a survey of three mills.

404. WARTMAN, FRANK S. Flotation tests on converter slag. Rept. of Investigations 3068, Bureau of Mines, 1931, 7 pp.
Flotation of ground converter slag containing 3 percent copper yielded 99 percent of the copper in 35 percent concentrates. Reagents used were pine oil, amyl xanthate, sulphuric acid, and P. E. oil.

405. WEIS, J. H. Granular and ground feldspar with a uniformly low iron content. Jour. Am. Ceram. Soc., vol. 14, 1931, pp. 413-418.
Discusses use of induction magnetic separators for removing iron from ground feldspar.

406. WELSCH, O. D. Chemical activity the cause of flotation? Eng. and Min. Jour., vol. 133, 1932, pp. 529-532.
Discusses the theory of flotation, suggesting that chemical reactions at the surface of mineral particles generate the gases which cause buoyancy. The air dissolved in the pulp is not enough to raise the particles, and that introduced into the cell aids only in bumping the particles to the froth.

407. WHEELER, ARCHER E., and EAGLE, H. Y. Katanga hydrometallurgical operations. Eng. and Min. Jour., vol. 133, 1932, pp. 143-144.
Reviews operations at the Katanga leaching plant during a period of low copper prices, indicating ability of the process as to capacity, extraction, acid and power consumption, and over-all costs.

408. WHITE, H. A. The absorption of gold in tube mills. Jour. Chem. Met. and Min. Soc. South Africa, vol. 31, 1930, pp. 161-171; vol. 32, 1931, pp. 294-298.
Though concentration upon the surfaces of pebbles and tube linings is a real effect, it does not account for more than a minor portion of the total gold locked up in the mill. The remainder is held in the numerous joints in the linings, between the lining and the shell and other crevices.

409. WHITE, H. M. The Champion Reef mine. Min. Mag. (London), vol. 47, 1932, pp. 73-82.
Describes in detail history, geology, development, mining methods, and treatment plant at the Champion Reef property in the Kolar district, State of Mysore, India. The ore is treated by a combination of blanket tabling and all-slime cyanidation recovering 98 percent of the gold. The stamp-mill blankets recover 44 percent, tube-mill blankets 34 percent, and cyanidation 20 percent. Includes flow sheets and descriptions of equipment.

410. WHITTENAU, E., and CRAMER, W. B. Milling methods and costs at the Copper Queen concentrator of the Phelps Dodge Corporation, Bisbee, Ariz. Inf. Circ. 6404, Bureau of Mines, 1931, 29 pp.
The Copper Queen concentrator at Warren, Ariz., was placed in operation in April 1923. The capacity is 4,000 tons of copper ore daily, but contemplated changes in the grinding circuit would increase it to 6,000 tons. The ore treated averages 2.15 percent copper, occuring as chalcocite with minor quantities of covellite, chalcopyrite, and bornite, associated with pyrite in a porphyritic gangue. Straight flotation is used to recover 91.32 percent of the copper in concentrates averaging 12.92 percent copper. Considerable pyrite is depressed in the lime, xanthate, pine oil circuit. The ore is reduced to all minus 1-inch in jaw, gyratory, and horizontal disk crushers, and ground in two stages, using rod mills and ball mills. Primary slime is separated in a bowl classifier treating the rod-mill discharge. The slimes, amounting to about one half the feed, are treated in separate flotation circuits and the sands are ground in ball mills. Forrester flotation machines are used in both the slime and sand circuits. The report describes the equipment used, gives detailed metallurgical and cost data, and discusses metallurgical and cost control.

411. WILLIAMS, CLYDE E. Iron ore beneficiation. Min. and Met., vol. 12, 1931, pp. 186-188.
Surveys methods and possibilities of application to Lake Superior ores. In 1930 over 13 percent of the ore of this district was beneficiated by washing, drying, and sintering and over 25 percent by crushing and screening. Beneficiation is increasing and should be increased further to conserve rich ores. Gaseous reduction to sponge iron is not recommended. The district will be a leader in production of iron ore long after the date of predicted exhaustion.

412. WILLIAMS, J. R. New process for treating antimony ores. South African Min. and Eng. Jour., vol. 41, part II, 1931, pp. 495-496.
Untreated antimony gold ore from the Murchison range, Transvaal, is not amenable to amalgamation or cyanidation. Special calcination volatilized the antimony and arsenic, then 74.44 percent of the gold was recovered by amalgamation and 15.66 percent by cyanidation.

413. WOAKES, RUSSELL B. India's only copper producer - the Indian Copper Corporation. Eng. and Min. Jour., vol. 132, 1931, pp. 51-53.
Describes mining, milling, and smelting at the Indian Copper property. Gives flow sheets of the concentrator.

414. WOODWORTH, SELIM E. Milling methods and costs at the Argonaut Mill, Jackson, Calif. Inf. Circ. 6476, Bureau of Mines, 1931, 12 pp.
The 300-ton Argonaut mill was built in 1916 at Jackson, Calif. About 81 percent of the gold in the ore occurs as coarse and fine metallic particles scattered through the quartz. Some of the gold is associated with pyrite and minor quantities of other sulphides including galena, sphalerite, chalcopyrite, tetrahedrite, and arsenopyrite. The ore is treated by a combination of amalgamation and gravity concentration; and the tailings, containing considerable carbonaceous or graphitic slimes, are treated under contract at a custom cyanide mill. Amalgamation recovers about 69 percent and concentration about 18 percent of the gold. The ore is reduced to 2½ inches in a jaw crusher and fed to 60 stamps crushing to all minus 28-mesh. Inside snd outside amalgamation is practiced. The plate tailings are classified in hydraulic classifiers and treated on vanners which remove concentrates. Vanner tailings are again classified, and the sands are reground in tube mills and re-treated on amalgamation plates and tables. Tailings flow by gravity to the custom cyanide plant.

415. YAMAGUCHI, KICHIRO. Studies on the mineral grains in relation to ore dressing. Jour. Faculty Eng. Tokyo Imp. Univ., vol. 19, 1931, pp. 263-388.
A detailed discussion of the methods of determining average particle size in screened mill products, with particular reference to the use of the microscope for this work. Points out erratic results of ordinary screening, and describes a new method of determining screen efficiency. Discusses effect of grain size and shape on ore-dressing operations.

416. YANCEY, H. F., and BLACK, C. G. The effect of certain operating variables on the efficiency of the coal-washing table. Rept. of Investigations 3111, Bureau of Mines, 1931, 13 pp.
Marked increase in efficiency was obtained by increasing the energy supplied by the head motion as represented by the total movement of the table deck in inches per minute (strokes per minute times 2 times length of stroke). For maximum efficiency of separation material should be discharged over all the commonly used portions of the periphery, 5 percent or less in the first foot nearest the head-motion end. Efficiency decreased with increase in feed from 3.5 to 7.5 tons per hour.

417. YANCEY, H. F., and PORTER, C. B. Pneumatic tabling of coal: Effect of specific gravity, size, and shape. Tech. Paper 536, Bureau of Mines, 1932, 18 pp.
In general, the size of the particles of a given specific gravity discharged from the wet table becomes successively finer with increasing distance from the mechanism end of the table, and in a given zone the average size of the components increases with increases in their specific gravities. With pneumatic tables the reverse conditions occur; the size of the particles discharged becomes increasingly coarser as the refuse end of the table is approached, and coarse coal is accompanied by finer impurities. With a wet table the most effective separation is made when the impurities are finer than the coal; with pneumatic tables the best results are obtained when the impurities are coarser than the coal. On the pneumatic table flat particles stratify at or near the top of the bed. In this position they are unsupported by the riffles and travel more rapidly than the cubical shapes across and down the transverse slope of the deck. Flakes concentrate in the zones nearest the head-motion end and cubical particles nearest the refuse end. This is the reverse of the wet table.

418. YOUNG, GEORGE J. Crushing and flotation at Morenci, Arizona. Eng. and Min. Jour., vol. 131, 1931, pp. 461-462.
Describes changes made recently in equipment and methods at the Morenci mill of Phelps Dodge Corporation, giving comparative metallurgical data representing periods before and after the changes were made. The mill is a combined gravity and flotation concentration plant, with tabling after the primary grinding and flotation after the secondary grinding. Many improvements are noted as a result of the changes. Includes flow sheets and metallurgical data.

419. _____ Cyaniding low-grade gold ore. Eng. and Min. Jour., vol. 131, 1931, pp. 561-563.
Describes operations at the cyanide plant of the Mountain Copper Co., Shasta County, Calif. The material treated is mined by shovels from the gossan of the old Iron Mountain pyrite deposit and contains about $2.00 a ton in gold. Crushing is to about 3/8-inch, and leaching is by percolation using very dilute cyanide solution. Very low operating costs are attained, due to the coarse cyanide feed.

420. YOUNG, GEORGE J. Gold-ore mining and milling. Eng. and Min. Jour., vol. 132, 1931, pp. 195-199.
Describes mining and milling at the Original Sixteen-to-One mine near Nevada City, Calif. Gives an historical sketch of mining operations and discusses the two amalgamation mills. The Sixteen-to-One mill, which treats 100 tons of ore a day, uses Hungarian riffles in the ball mill-classifier circuit and stationary tables on the classifier product. Riffle products are treated in a Berdan pan. Includes flow sheets.

421. ____ Selective lead-zinc flotation at Kimberley, British Columbia. Eng. and Min. Jour., vol. 131, 1931, pp. 313-315.
Detailed description of milling operations at the Kimberley mill of the Consolidated Mining & Smelting Co. of Canada (Ltd.). For a discussion of the mine see pages 215-217. Jaw and gyratory crushers at the mine reduce the ore to about 2-inch and a cone crusher and rolls at the mill reduce it to 98 percent through $\frac{1}{2}$-inch. Grinding is done in three stages in ball mills in closed circuit with large classifiers, and flotation in Minerals Separation machines, using soda ash, sodium cyanide, oil mixture (water-gas tar, creosote, and cresylic acid) and xanthate in the lead circuit and copper sulphate, sodium bichromate, cresylic acid, and oil mixture in the zinc circuit. About 90 percent of the lead is recovered in lead concentrates and 83.5 percent of the zinc in zinc concentrates. All concentrates are shipped to Trail, British Columbia, for reduction. Article gives flow sheets and metallurgical data.

422. ZEIGLER, W. L. Milling in the Coeur d'Alene district, 1930. Min. and Met., vol. 12, 1931, pp. 88-89.
Briefly reviews milling operations at Coeur d'Alene plants during 1930, with short descriptions of the Jack Waite and Sunshine mills. Describes mill equipment in general use and discusses flotation reagents. All mills in the district now regrind and float the gravity section tailings.

423. ____ Milling methods and costs at the lead concentrator of the Hecla Mining Co., Gem, Ida. Inf. Circ. 6600, Bureau of Mines, 1932, 16 pp.
Detailed description of a 750-ton concentrator treating an ore containing argentiferous galena, with minor amounts of zinc sulphide in a quartzitic gangue. The method employed is a combination of jig concentration followed by flotation of the reground jig tailings, with direct flotation of primary slime and reground jig middlings. Report gives detailed flow sheets, metallurgical tables, cost data, flotation reagents, recoveries, etc.